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REPORT ON
CANADIAN GOLD MINES
WITH SPECIAL EMPHASIS ON
MILLING OF REFRACTORY ORES

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ACKNOWLEDGEMENT

The writer wishes to acknowledge the whole-hearted co-operation and assistance received from gold mill operators and staff specialists who provided much of the information presented in this report.

INTRODUCTION

As authorized by Mr. E. G. Joe, the writer made a detailed study of Canadian gold mining operations, with special emphasis on the treatment of refractory gold ores. This study is based on information obtained from published reports, from notes of visits to most of the major gold producers, and from other personal communications. Use has been made of earlier reports to CANMET on visits to gold mines in the Northwest Territories and to the Dome Mine.

In addition to gold producers, two of Canada's largest silver operations have been included in this survey. One of these, Equity Silver Mines also produces gold. Gold mines which produce gold in concentrate form along with copper or other metals, have not been included.

Mines surveyed include all known significant producers in 1984 (some have subsequently shut down), imminent producers in the Hemlo area and some of the prospective producers across Canada. With the exception of Hoyle Pond, none of the prospective mines is committed to go into production.

Gold production for 1984 from the mines included in this survey are estimated to be as follows:

	Ounces
Quebec	570,000
Ontario	766,000
British Columbia	72,000
Northwest Territories	407,000
Total	1,815,000

British Columbia production is a rough estimate and excludes production from Equity Silver.

Agnico-Eagle's silver division and Equity Silver Mines produced 5,730,000 ounces of silver in 1984 in the form of concentrates.

This report has been prepared for the sole purpose of assisting CANMET in directing its research efforts into the most fruitful areas of investigation.

TREATMENT OF REFRACTORY ORES

The most refractory gold ores presently being milled in Canada are arsenical ores in the Yellowknife and Red Lake areas, in which gold occurs in sub-micron form in arsenopyrite.

At Giant Yellowknife, the arsenopyrite is recovered by flotation and roasted in two stages. Roaster calcine and dust are cyanided separately to recover gold. At least 70% of the gold in the ore would not be recovered if the arsenopyrite were not roasted. Even with an efficient roasting plant, about 14% of the gold is not recovered. Gold losses are about equally divided between the flotation tailings, which are not worth leaching, and the leached residues from roasting.

At Campbell Red Lake about 15% of the gold would be lost if the arsenopyrite were not roasted. However, by roasting, and cyaniding the flotation tailings and roaster products Campbell recovers 94.5% of the gold in the ore.

At Dickenson, roasting has been abandoned and approximately 84% of the gold is recovered by amalgamation and cyaniding the total ore. An additional 5% is recovered in a flotation concentrate which is shipped to Boliden, in Sweden. With an efficient roasting plant Dickenson could recover about 95% of the gold in the ore.

The presence of reactive carbonaceous material, probably graphite, in mill feed is a problem in several mines. The reactive material acts as a precipitant for gold in cyanidation. At Kerr-Addison and Agnico-Eagle, problems are minimized by adding kerosene or fuel oil to the curcuit to coat the graphite and render it inert during cyanidation. At other mines, including Schumacher and Owl Creek, in the Timmins area, most of the active carbon is floated with a copper concentrate which goes to a smelter. However, some difficulty has been reported in floating all the carbon and, at the same time, in maintaining an acceptable grade of copper concentrate. At Carolin, the reactive carbon was allowed to go through the cyanidation circuit, after which it was recovered by flotation and treated by some type of batch process to recover the gold associated with it.

Finely disseminated gold, both in ganque minerals and sulphides is common in Canadian ores and many ores are very hard to grind. As a result, the cost of grinding to liberate the gold is a major factor in milling costs. In some ores the gold occurs primarily in sulphides, particularly pyrite, which may be liberated at a relatively coarse grind. By floating the sulphides and regrinding the concentrate only, grinding costs can be reduced. This procedure is followed at Parmour's No. 1 mine. However, because the mill feed and flotation tailings are low in grade, the tailings are not worth cyaniding. Gold recovery is only 88.5%.

Many mines report the presence of gold tellurides in some ores but concentrations are apparently small and gold losses haven't been attributed to tellurides.

GENERAL OBSERVATIONS

There are numerous variations in gold milling processes, reflecting the geological diversity of gold occurrences as well as personal preferences.

Apparently due to lack of confidence in the long-term price of gold the mines have been slow to modernize their plants and, when carried out, it has usually been in piece-meal fashion.

Capital and operating costs in new plants have been reduced by going to larger equipment, locating more equipment outdoors, using more efficient equipment and, possibly, by more efficient metallurgical processes.

At least five new mills are using carbon-in-pulp to extract gold from solution but most haven't operated long enough to make a definitive comparison with the Merrill-Crowe process. Several new mills also use semi-autogenous grinding. Some companies are still opting for conventional crushing and grinding circuits, citing operating problems in some of the semi-autogenous circuits and the inherently higher power efficiency of conventional circuits. Many small operators are influenced in process selection by the availability of used equipment.

Only one recent plant with new equipment selected air-lift agitators. High efficiency mechanical agitators are the popular choice, with supple-

mental air lines to ensure adequate aeration. Some plants are testing Atara "Cannon" mixers.

There is renewed interest in gravity concentration, particularly as a replacement for amalgamation, which will soon be phased out of all gold mills.

There is no great interest in instrumentation, particularly in cyanidation. For the most part, operating conditions are not critical and the long retention time in leaching gives operators plenty of time to respond to changes in mill feed.

Most operators have developed operating parameters through empirical methods rather than from a knowledge of solution chemistry. Great emphasis is placed on maximizing recovery, possibly at the expense of high reagent additions and high grinding costs.

Many ores contain pyrrhotite or other reactive sulphides that are attacked by cyanide solution, resulting in high cyanide consumption and in the formation of cyanide compounds that inhibit gold extraction and aggravate environmental problems. When reactive sulphides, particularly pyrrhotite, are present in appreciable concentration, it is common practice to aerate the slurry prior to cyanidation, and in some cases, to add litharge or lead acetate to the leaching circuit to reduce cyanide consumption and/or to increase gold recovery. Operating practice with ores containing pyrrhotite vary considerably from mine to mine; there is a general lack of knowledge of the chemistry involved and generally poor correlation between laboratory and plant testing.

Some mines are using activated carbon to scavenge gold from waste ponds and one mine, Dickenson, is installing a carbon-in-pulp circuit to scavenge gold from plant tailings slurry.

Two plants have Tower mills in operation. At Macassa a mill is in regular service. At Cominco's Con mill a Tower mill was being tested in 1984. Both mills have been used for fine grinding of concentrate and both operators report lower power consumption, compared with ball milling, but very high wear rates in the Tower mills. New materials are being tested in an attempt to reduce wear to acceptable levels.

Most mines presently use natural degredation in large ponds to destroy cyanide compounds in mill wastes, but some mines will probably be required to add chemical destruction systems in the future. There is concern regarding the cost of these systems, particularly for chemicals.

Most small gold mines in Canada are in financial difficulty and some reported in this survey are now closed, pending an increase in the price of gold.

RECOMMENDATIONS

Several areas of research are recommended for CANMET'S consideration.

Some arsenical ores in Canada, e.g. Dickenson Mines, contain pyrite as well as arsenopyrite but the concentration of gold in the pyrite is quite low. It could benefit Dickenson and possibly some potential producers if more effective methods of separating pyrite from arsenopyrite were developed. Inclusion of barren pyrite in concentrates treated by roasting or oxidative pressure leaching results in substantially higher costs of treatment.

Further research into the chemical reactions involved in the treatment of gold ores containing pyrrhotite could result in decreased cyanide consumption and in lower formation of cyanide compounds that can inhibit gold leaching and contribute to environmental problems. Specific areas deserving investigation include leaching at lower temperatures and magnetic separation of pyrrhotite from ores prior to cyanidation.

Many mines have invested large sums of money in providing tailings ponds and lagoons with the expectation that natural degredation of cyanide complexes will reduce cyanide levels to acceptable concentrations. Methods of promoting natural degredation warrant investigation. Further research in methods of regeneration of cyanide from waste liquors is recommended.

SECTION I

OPERATING MINES

AGNICO-EAGLE GOLD DIVISION

Ownership

The operating properties of the Gold Division consist of the original Eagle Mine and the contiguous 97% owned Telbel Mine.

Location

Four miles northwest of the town of Joutel in Northwestern Quebec.

Economic Metals

Primarily gold, with some silver.

Ore Reserves

Total proven reserves, December 31, 1983 were 296, 357 tons at an average grade of 0.188 oz. of gold per ton.

Total proven probable and drill-indicated reserves at December 31, 1983, including a 35% allowance for dilution, were 1,184,427 tons at an average grade of 0.202 oz/ton.

Operating Costs

For nine months ending September 30, 1984, costs were C $263 per ounce of gold produced.

History

The Eagle Mine has been in operation since October of 1974. Problems with low gold extraction, high reagent consumption and poor filtration were solved and the plant capacity was raised in stages from 550 TPD in 1974 to the present capacity of 1150 TPD.

Originally ore came from the No. 1 shaft, which was developed to 2850 ft. The new Telbel shaft, 4200 ft. east of the No. 1, was completed recently to its initial objective of 4000 ft. Most of the mill feed now comes from the Telbel shaft, which is on the downward extension of the original orebody.

Geology and Mineralogy

The major ore type consists of fine-grained massive pyrite mineralization associated with highly carbonaceous altered dacite, slaty sedimentary rocks and 5 - 10% quartz and carbonate material. The other type of ore is represented by coarse, euhedral pyrite crystals in highly chloritic rock. Gold grains are disseminated, generally smaller than 5 microns in the various types of pyrite. There is a minor amount of pyrrhotite in the ore and it also contains gold. Some graphite appears in the ore at times from mine dilution. Low concentrations of magnetite, chalcopyrite and arsenides are found in certain parts of the mine.

Ore from the lower levels of the mine contains higher concentration of gold, but less silver, than ore from the upper levels.

Mining Methods

Most of the ore is mined with long hole stoping and the remainder by shrinkage stoping.

Annual Tonnage Milled/Ore Grade

During the nine months ended September 30, 1984, the mill treated 282,000 tons at an average rate of 1,088 tons per operating day. Operating time was 95%. Average grade was 0.180 oz/ton gold.

Gold Recovery %

Mill recovery averaged 90.45% for the 9-month period.

Gold Production

Gold and silver productions for the same period were 45,944 oz. and 7,172 oz. respectively.

Milling Process

The flowsheet has been changed considerably in the last few years. The present flowsheet is shown in the Appendix. It consists essentially of the following:

Conventional crushing and grinding to 70% - 400 mesh
Rougher flotation using Maxwell and Denver cells (Tails to waste)
Rougher concentrates (1/3 of mill feed) reground to 90% - 400 mesh
Preaeration in pachucas, using lime only, at 23% solids
Primary cyanidation at 23% solids
Thickening to 60% solids (overflow to precipitation)
Secondary cyanidation at 58% solids
Secondary thickening (with wash) to 60% solids
Two-stage vacuum drum filtration
Gold recovered from pregnant solution by zinc precipitation
Precipitate recovered in presses and smelted to bullion
Tailings, consisting of leach residue and flotation tailings pumped to a 60-acre tailings pond.

At present effluent from tailings overflows to a 25 acre polishing pond. Cyanide degrades naturally in the ponds.

A new, larger tailings pond now under construction is scheduled to be completed this year.

Occasionally fuel oil is added in flotation to render graphite inert to gold values during cyanidation.

Operating Data

Leach retention time - primary 6 hr., secondary 48 hr.	
Lime in preaeration and cyanidation	1.0 lb/ton solution.
Sodium cyanide in cyanidation	1.5 lb/ton solution.
Cyanide consumption	1.8 lb/ton of mill feed.

Refractory Ore

All the ore is somewhat refractory to cyanidation, requiring extremely fine grinding to liberate gold in pyrite.

Comments

The Eagle mill is one of the few in Canada to float the gold with pyrite and cyanide only the concentrate. In 1978 it was reported that only 1.64% of the gold was lost in the flotation tailing.

Preaeration and primary leaching at very low density is unusual. Low density probably results in more effective aeration and reduced concentrations of cyanicides. No lead salts are used in preaeration.

Outlook

The new shaft is expected to extend the mine life for many years. Currently the mill is being expanded to 1800 tons/day.

References

1. Mikulash, K. O., Milling Practice in Canada, 1978. Pp 49, 50.
2. McQuiston, F. and Shoemaker, R. S., Pp. 139-142.
3. Agnico-Eagle Mines Ltd., Annual Report, 1983 and Interim Report for period ending September 30, 1984.
4. Personal communication, Gaetan Goulet.

BACHELOR LAKE GOLD MINE

Location

About 3 miles east of Desmaraisville, in Lesueur Township, Northwestern Quebec.

Ownership

In March 1984 Quebec Sturgeon River Mines held a 53% interest in Bachelor Lake. Both mines are part of the Coniagas group.

Economic Metals

Gold.

Ore Reserves

At December 31, 1984; mineable reserves above the 7th (approximately 1,000 feet) level were estimated to be 244,000 tons at an average grade of 0.190 oz. gold per ton. Geologic reserves between the 7th and 12th levels are estimated at 336,000 tons averaging 0.200 oz. per ton. Reserves include a 20% dilution factor.

Operating Costs

Mining, milling and camp costs, excluding depreciation and amortization, were $8,546,177 for the full year of 1984, equivalent to $425 per ounce of gold produced.

History

Milling commenced in June, 1982. Operations in 1984 were adversely affected by a shaft sinking program which limited access to certain stope areas of the mine with a resultant decline in both tonnage and grade of ore treated in the period from April to December.

Shaft sinking, originally targeted to go to 1,700 feet was suspended in January, 1985 after reaching the 9th level. It is hoped that it will resume later this year.

Geology and Mineralogy

The property is underlain largely by a series of interbedded agglomerates and tuffs of basic to intermediate composition with minor felsic volcanics. The mineralized zone occurs in a cross-cutting fracture dipping about 55°. It consists of multiple quartz veins in a silicified and mylonitized reddish rock.

Mining Methods

Long hole methods and shrinkage stoping.

Annual Tonnage Milled/Ore Grade

For the full year of 1984, 156,086 tons were milled with an average grade of 0.140 oz/ton. Mill feed included some shaft waste, abnormal dilution from some mining areas as well as some low grade stockpiled ore.

Gold Recovery %

Gold recovery averaged 92.4% in 1984. Emphasis directed toward an improvement in mill recovery was successful. Recovery, which averaged about 90% early in 1984, attained 94% in December. A third filter, scheduled to be operational early in 1985, was expected to improve recovery further.

Gold Production

For the full year of 1984, 20,104 ounces of gold were produced.

Milling Process

The conventional cyanidation mill includes the following:

Crushing - jaw crusher and short-head cone
Grinding - two stages of closed circuit ball-milling
Thickening
Primary leaching - 5 stages
Secondary leaching - 1 stage
Wash filtration - 3 stages, drum filters
Merrill-Crowe circuit
Smelting to produce bullion
Waste pumped to tailings pond with capacity for 1.2 million tons.

A complete flowsheet is presented in the Appendix.

Leaching tanks are equipped with Prochem agitators. Retention time is 36 to 40 hours in the primary agitators. The secondary agitator is probably used as a surge tank.

Refractory Ore

Refractory ore has not been reported on the property.

Outlook

Appears to be a marginal operation at current price of gold.

References

1. Hawkins, T. Canadian Mining Journal, May, 1982, Pp. 39-45.
2. Canadian Mines Handbook, 1984-85, P. 46.
3. Bachelor Lake Gold Mines Annual Report, 1984.

BELMORAL MINE

Location

The Ferderber Mine and mill are located 6 miles north-east of Val d'Or in northwestern Quebec. The Dumont Mine is 2 miles south-west of the Ferderber.

Ownership

Belmoral Mines Ltd. owned the Ferderber Mine and mill and Bras d'Or Mines Ltd.'s. G.H. Dumont mine. Bras d'Or Mines Ltd. had a 45% net proceeds of production interest from the Bras d'Or mine. On July 12, 1982 Continental Illinois Bank (Canada) of Toronto seized Belmoral's assets after calling in a $30 million loan. Continental have continued the mining and milling operations.

In October, 1984 Boliden Canada Ltd. announced tnat it had signed a letter of interest to purchase the assets of Belmoral Mines Ltd. from Continental.

Economic Metals

Gold.

Ore Reserves

Current figures not available.

Operating Costs

In 1983 operating costs, including depreciation and amortization, were $410 per ounce of gold produced.

History

The Ferderber Mine was brought into production in 1979 and the Bras d'Or in 1980. The Ferderber mine experienced a tragic cave-in in May, 1980 but resumed production on Nov. 3rd of the same year. Initially the ore was custom-milled at Louvem's plant in the area. In November, 1981 Belmoral brought their own mill into operation.

Geology and Mineralogy

Several mines in the Val d'Or area are located along the fringe of the Bourlamaque granodiorite or within volcanics close to the border of the batholith. The Ferderber is the first mine located well within the batholith to go into production. The Bras d'Or mine is granodiorite close to the edge of the batholith.

The gold bearing structures consist of quartz veins with pyrite and occasional local chalcopyrite with fine gold associated with the pyrite. The veins occur in a strong steeply dipping shear zone, shown to be quite continuous,with widths ranging from 10 to 50 ft.

Where the shear zone consists of schist without any quartz, values are almost always low although usually miniature lenses of quartz with pyrite can be found. The values are essentially associated with the pyrite and are directly related to the percentage of pyrite.

Mining Methods

Shafts have been completed to the 1,250 ft. level on the Ferderber and to the 1,450 ft. on the Bras d'Or. Long hole stoping and shrinkage mining methods have been used.

Annual Tonnage Milled/Ore Grade

Recent data are not available.

In the 44-week period ended May 13, 1984, 165,944 tons were milled at an average head grade of 0.170 ounces per ton (apparently recovered grade).

Gold Recovery, %

For the 44-week period referred to above gold recovery averaged 93.06%.

Gold Production

Gold produced in this period amounted to 28,150 ounces. For 28 weeks ending January 20, 1984, some 32,600 ounces were reported to have been produced.

Milling Process

Believed to be straight cyanidation with zinc precipitation.

Refractory Ore

No refractory ore is reported.

Outlook

Belmoral went bankrupt because of a heavy load of debt. Boliden must have confidence that the mine can be profitable. There are several potential ore zones on the property that haven't been thoroughly investigated.

References

1. Bergmann, H.J. The Belmoral story -- persistence pays off. Can. Min. Journal, April, 1980, Pp. 29 - 32.
2. Belmoral Mines Ltd. Annual Report, 1983.
3. The Globe and Mail, October 18, 1984, P. B 12.
4. Canadian Mining Journal, February 1981, P. 102.

LA MINE BOUSQUET

Location

The Bousquet mine is located in Bousquet Township near Cadillac in northwestern Quebec. It is about 3 km. east of the Doyon mine.

Ownership

La Mine Bousquet is a division of Lac Minerals Ltd.

Economic Metals

Gold.

Ore Reserves

No. 1 and No. 3 zones are underground operations. No. 5 zone is a lower grade open-pit operation.

Proven and probable ore reserves from the various zones at December 31, 1984 were as follows:

	Tons (1000's)	Ounces of Gold per ton
Proven		
Zones 1 and 3	4,637	0.140
Zone 5	4,914	0.060
Probable		
Zones 1 and 3	1,770	0.140
Zone 5	1,354	0.070

Zone 5 reserves are processed only when mill capacity is available.

A dilution factor of 20% grading 0.02 ounces per ton has been included in the calculation of proven reserves in Zone 5.

History

The Bousquet mine was discovered by Lac's Exploration Division in 1977. The operation, which consists of an underground and a low-grade open-pit mine, began production in 1979. A shaft has been developed to 1,858 ft.

Since that time Bousquet ore has been processed in the Est Malartic or Terrains Auriferes (Malartic Goldfields) mills. In 1984 Bousquet ore was the predominant feed for the Terrains mill. This mill went into operation at 400 tons per day in 1939. By 1952 the rate had been increased to 1,900 tons per day.

Geology and Mineralogy

The Bousquet ore is a pyritic deposit that lies in the Abitibi greenstone belt. Ore zones are of stratabound volcanogenic origin.

There are eight gold-bearing zones. The most important zone, No. 3, is 3 m. to 30 m. wide and gold values are associated with felsic tuffs. High gold values are found in a narrow, massive to semi-massive pyrite horizon. Lower gold values are disseminated through the ore zone. In another zone, No. 1, gold values are associated with pyrite and chalcopyrite stringers within a banded tuff horizon.

Gold occurs in the native state and to a minor extent in the telluride minerals, petzite and calaverite. Besides pyrite, which has been reported to make up 8% of the ore, there are minor concentrations of sphalerite, galena, tetrahedrite and altaite (lead telluride). Gangue constituents include quartz, muscovite, chlorite, carbonaceous material and sphene.

Mining Methods

Some ore comes from a low grade open pit, but most is produced underground, from Zones No. 1 and No. 3.

Ore has been mined mainly using a modified sub-level caving method. In a few areas flat back cut-and-fill and sub-level cut-and-fill have been used.

The orebody and immediate country rocks are sericitic and highly schistose. Poor ground conditions have resulted in mining problems which have generally been overcome.

Operating Costs

Average cash operating cost per ounce of gold produced in 1984 was $289.

Annual Tonnage Milled/Ore Grade

In 1984, 589,000 tons were milled with an average grade of 0.182 ounces per ton.

Gold Recovery

Gold recovery averaged 92.9% for the year.

Gold Production

In 1984 gold production was 99,425 ounces.

Milling Process

The Terrains Auriferes (Malartic Goldfields) mill employed a straight cyanidation process. The circuit included 3-stage crushing, 3-stage grinding, cyanidation, 2-stage filtration and gold recovery by the Merrill-Crowe process. Solid tailings were discarded or sent to a backfill plant. Apparently few changes were made to the mill to treat custom ores after the Malartic Goldfields mine ceased production.

Operating Data

Grind to 85% - 200 mesh.

Leach retention time is 30 hr.

Lime concentration in leaching 1.5 lb./ton of solution.

Cyanide concentration in leaching 1 - 1.5 lb./ton of solution.

Reagent consumptions:

Grinding media	1.9 lb./ton
Lime - approx.	4 lb./ton
Calcium cyanide	1.2 lb./ton (NaCN equiv.)
Lead nitrate	0.1 lb./ton
Zinc dust	0.05 lb./ton

Refractory Ore

The ore is not refractory. Gold losses in the mill are apparently due to the fine dissemination of gold.

Comments

Lime concentration in leaching is quite high, apparently to ensure extraction of the tellurides.

Outlook

With generous ore reserves and reasonable operating costs Bousquet will probably operate for at least several years, however, mining problems with poor ground could become more serious at lower levels.

References

1. The Milling of Canadian Ores, CIM Congress Vol. 1957, Pp. 113-116.
2. CIM Bull. Jan. 1985, Pp. 35-40.
3. Geology of Canadian Gold Deposits, CIM Spec. Vol. 24, P. 29.
4. Lac Minerals Ltd. Annual Report, 1984.
5. Conversations with George Hope and J. McMullen.

CAMFLO/MALARTIC HYGRADE MINES

Location

The Camflo mine and mill are located 8 km east of Malartic, in Northwestern Quebec. The Camflo ore zone extends into the adjacent Malartic Hygrade property.

Ownership

Camflo Mines Ltd. merged with Barrick Resources Corp. in 1984, with Barrick ending up with 57% interest in the new company, Barrick Minerals (Canada) Inc.

Economic Metals

Gold

Ore Reserves

On December 31, 1983 total proven and indicated reserves on Camflo's property were 2,100,000 tons averaging 0.12 oz. gold per ton. On the property of Malartic Hygrade Gold Mines, reserves were 1,119,000 tons averaging 0.129 oz. gold per ton, in the main orebody, open to depth.

Operating Costs

Malartic Hygrade reported that operating costs for the 9-months period ending September 30, 1984 were C $201 per ounce. (Camflo mines and processes The Malartic Hygrade ore under a production sharing agreement.)

History

The Camflo mine went into production in 1965 at 1,000 tons per day capacity. Initially, the ore was custom-milled at Malartic Goldfields to test its amenability to cyanidation. The 1,000 ton per day Camflo mill went into operation on the mine site in July, 1969. In 1975 the capacity of the mill was increased to 1,250 tons per day. Regular production from the Malartic Hygrade mine began in December 1981.

Geology and Mineralogy

The Camflo orebody is 2 miles north of a major break known as the Cadillac-Bouzan fault and is associated with a porphyritic monzonite stock.

Most of the ore (about 90%) has been found within the stock of porphyritic monozonite in both large and small irregular zones. The ore zones make up about 25% of the porphyry and the rest is barren. The gold-rich ore is almost identical to the barren porphyry, except that it contains 3 to 5% white-to-grey quartz in stringers and 1 to 3% coarse disseminated pyrite. Small specks of visible gold are often found on the faces of the pyrite crystals and in the quartz stringers. Minor amounts of scheelite and tellurides occasionally are found in the higher-grade ore.

The balance of the ore, diorite ore, is found along fault zones associated with minor amounts of fault-zone volcanics. It contains about 50% quartz as stringers varying in width from 1 to 10 ft. and containing 1 to 3% pyrite. The gold content varies roughly with the pyrite content and ranges from 0.01 to 0.40 oz/ton.

The two types of ore are blended for milling.

Mining Methods

Over half the ore is mined using longhole open stoping and the balance by shrinkage stoping and a modified room and pillar method.

Annual Tonnage Milled/Ore Grade

For the full year 1983 ore milled from both properties amounted to 466,389 tons with an average grade of 0.176 oz./ton (calculated grade assuming 95% recovery).

Gold Recovery, %

Gold recovery currently is at least 95%.

Gold Production

Total production from both properties amounted to 77,936 oz. for the full year 1983.

For the 9-month period ended September 30, 1984, gold production was down 30% from the same period in 1983 due to unexpected localized dilution in one area of the mine, and the overall mining sequence. The situation was not expected to continue.

Milling Process

The mill flowsheet is unchanged from that shown in Milling Practice in Canada in 1978. (3 - 38' x 14' thickeners shown are double deck).

Essentially the process consists of:

Two-stage cone crushing.

Rod and ball-mill grinding, in cyanide.

Thickening of ground product.

3-stage cyanide leaching, with drum-filtration following each stage.

Zinc precipitation and pressure filtration.

Smelting to recover bullion.

Tailings pumped to large pond where natural degredation of cyanide occurs.

Operating Data

Refer to notes of plant visit in the Appendix.

Refractory Ore

Ore is not refractory and gold recovery has been 95-96% since the mill started up.

Comments

The mill design is compact but provides adequate access for maintenance. It is regarded as one of the best of the "conventional" gold mills in Canada using crushing, grinding, straight cyanidation and zinc precipitation.

Outlook

With low operating costs and fairly large ore reserves an extended period of operation is likely from the two properties.

References

1. Tour Stop - Camflo Mines Limited, CIM Reporter, February, 1976.
2. Camflo Mines Limited, CIM Reporter, September 21, 1980, Pp. 6-8.
3. Camflo Mines Limited, Milling Practice in Canada, 1978.
4. Northern Miner, December 6, 1984, P. A2.
5. Notes of Visit to Camflo Mine, March 25, 1985 (See Appendix).
6. Canadian Mines Handbook, 1984-85, P. 71.

LOUVEM'S CHIMO MINE

Location

The mine is located in the Val d'Or area of Northwestern Quebec.

Ownership

The mine is owned by Societe Miniere Louvem, a subsidiary of SOQUEM.

Economic Metals

Gold

Ore Reserves

At year-end 1984 reserves in two zones were reported to be approximately 950,000 tons with an average grade of 0.25 ounces per ton.

Operating Costs

No information.

History

In 1966-67 the mine shipped 142,000 tons with an average grade of 0.40 ounces per ton, to the Bevcon mill. In August, 1984 Louvem reopened the mine after deepening the shaft from 135 m to 335 m and opening up four new levels. In addition the company completely rebuilt their Manitou mill to process gold ore. Louvem leases the mill from SOQUEM. Operations were suspended in January, 1985 for 5 months in order to develop a new ore zone.

Geology and Mineralogy

Free gold occurs with arsenopyrite and pyrrhotite in quartz-carbonite in shears and fractures at the contact between a diorite sill and an iron formation.

Mining Methods

No information.

Annual Tonnage Milled/Ore Grade

The mill is rated at 300 tons per day but has only operated for 5 months. The average grade of ore in the original zone was 5.90 g/t. (0.17 oz/t.).

Gold Recovery, %

No information.

Gold Production

Assuming that the mill operated steadily for 5 months at 300 TPD, a diluted feed grade of 0.15 ounces per ton, and 90% recovery, the mill may have produced 6,100 ounces of gold in 1984.

Milling Process

Probably a conventional straight cyanidation process but may have floated the gold and cyanided the concentrate.

Refractory Ore

It is unlikely that the ore is refractory.

Outlook

The company is hopeful that a new, higher grade zone of ore discovered 500 ft. south of the shaft, can be profitably mined and operations resumed in 1985.

References

1. Mineral Bulletin MR 198, Energy Mines & Resources Canada, P. 57.
2. CMJ Dec. 1984, P. 11.
3. Northern Miner, Jan. 3, 1985, P. A3.

DEST-OR MINE

Location

The mine and mill are located in Destor Township, approximately 40 km north of Noranda, Quebec.

Ownership

Aiguebelle Resources Inc. (90%), Q.M.G. Holdings Inc. (10%).

Economic Metals

Gold and silver.

Ore Reserves

Late in 1984 proven, probable and possible reserves in the main zone were 2.4 million tonnes averaging 0.14 oz. gold per ton, including a 20% dilution factor. This excludes potential reserves north and south of the mine workings.

Operating Costs

In early 1985 production costs were about $290 per ounce, including depreciation. The company expects costs to be in the $225-$250 per ounce or $40 per tonne when a new shaft is in operation.

History

Mine production commenced in late 1982. Until Dest-Or's own mill came into operation in July 1984, the ore was trucked to Louvem's Chimo mill, some 160 km to the east, for custom milling.

Dest-Or's mill is now operating satisfactorily and a new shaft is being developed to replace a ramp that now provides access to the mine.

Nominal mill capacity is 1,100 tons per day.

Geology and Mineralogy

The ore contains some pyrite but no other sulphides. Gold is very finely disseminated, mostly finer than 10 microns. Pyrite grains are mostly less than 20 microns.

Mining Methods

No information available.

Dilution has been higher than expected.

Annual Tonnage Milled/Ore Grade

For nine months ending December 31, 1984, mills processed 177,500 tons, including 26,540 tons of low-grade ore containing 0.08 oz/ton. Average grade of ore milled was 0.127 oz/ton gold.

Gold Recovery %

Gold recovery averaged 91.3% in December, 1984.

Metal Production

Gold and silver production for the same nine-month period was 18,166 oz. and 6,382 oz. respectively. In the third quarter of fiscal 1984 (ending December 31) 9,717 oz. gold and 3,483 oz. silver were produced.

Milling Process

The milling process is simple, consisting of the following:

Jaw crushing run-of-mine ore to 4 in.

Semi-autogenous and ball mill grinding, without cyanide.

Thickening of cyclone overflow from ball mill circuit.

Cyanide leaching (4 tanks).

Gold adsorption in 5 carbon-in-pulp stages.

Atmospheric stripping - 2 columns in parallel.
Gold deposition on cathodes by electrowinning.
Induction furnace smelting of cathodes to produce bullion.

Tailings are pumped to a pond that will take two years to fill up.

Operating Data

Ball mill grind 90%-200 mesh
Cyanidation retention time 28 hr. (complete in 16 hr.)
Cyanide concentration 1.5 lb./ton of solution, NaCN equiv.
Cyanidation pH 10.5-11.0 (single lime addition to grinding)
Stripping - atmospheric, 95°C, 3 days using 1% NaOH, 0.3% NaCN

Reagents:		
	Calcium cyanide	2 Kg./tonne (1 Kg/t NaCN equiv.)
	Lime	0.5 Kg./tonne
	Lead salt	small quantity in grinding
	Carbon	700 Kg./mo. estimate

Refractory Ore

All the ore is somewhat refractory due to the fine dissemination of gold, requiring very fine grinding to liberate the gold.

Comments

Dest-Or is unusual in that it contains no sulphide minerals apart from pyrite. The effective leaching period of 16 hours is unusually short.

The mill is one of a new generation of gold mills. Capital costs were saved by using semi-autogenous grinding, by locating the thickener and leach tanks outside, and by using carbon in pulp for gold recovery.

Total number of operating and maintenance personnel is only 17 excluding assayers.

References

1. Kroon, A. S. Northern Miner, June 21, 1984, p. B4
2. Northern Miner, Oct. 11, 1984, Pp. 1, 2, 3.
3. Envers, Peter, Canadian Mining Journal, Nov. 1984, Pp. 51-53.
4. North American Gold Mining Industry News, March 15, 1985, Pp. 1, 2.
5. Mill visit by I. C. Edwards, March 19, 1985.

LA MINE DOYON JOINT VENTURE

Location

La Mine Doyon is located in the Abitibi district some 45 km. east of Rouyn in Northwestern Quebec.

Ownership

Lac Minerals operates and shares a 50% interest in La Mine Doyon Joint Venture with SOQUEM (Societe Quebecoise d'Exploration Miniere).

Economic Metals

Gold.

Ore Reserves

Proven open pit reserves at year-end 1984 were 1,940,000 tons with an average grade of 0.137 ounces per ton. Probable underground reserves amounted to 2,895,000 tons averaging 0.184 ounces per ton. Open pit reserves include a 10% dilution factor. Underground reserves were calculated without a dilution factor.

Additionally low grade ore, amounting to 635,000 tons with an average grade of 0.051 ounces per ton, is stockpiled on surface.

Operating Costs

Average cash operating cost in 1984 was $214 per ounce of gold produced. This includes cost of waste and overburden removal but excludes non-cash items (presumably depreciation, etc.).

History

The operation began production as an open-pit gold mine in March, 1980. In 1983, a declined access ramp was started to allow underground exploration to determine the possibility of underground mining. The first phase of underground development was completed in February, 1985 with completion of a headframe and surface plant and some initial underground development work. The shaft is planned to be completed to a depth of 2,100 feet.

Initially the ore was custom-milled off-site. Mill production began on-site in November, 1982 and by early 1983 the mill was operating at 1,500 tons per day. Custom milling of some ore has continued. In 1984, 55% of the ore mined was milled off-site.

Geology and Mineralogy

There are three major rock types at La Mine Doyon, namely:

1. Grey agglomerate.
2. Green agglomerate.
3. Sericite Schist.

The first type, the grey agglomerate, a felsic volcaniclastic, makes up about 80% of the mineralized zone. The gold mineralization is associated with centimeter wide veins of quartz with carbonate, pyrite, chalcopyrite and visible gold.

The second rock type, green agglomerate, a mafic volcaniclastic, constitutes about 10% of the mineralized zone. Gold mineralization is associated with veins of pyrite, quartz and chalcopyrite. Pyrite content is typically 2 to 5%.

The third group, sericite schist, of the feldspar group, makes up the remaining 10% of the mineralize zone. The gold mineralization is associated with concentrations of pyrite in veins forming from 10 to 30% of the rock.

There are two principal types of gold association:

1. Fine grains disseminated within the pyroclastic rocks along silicate and pyrite grain boundaries and as inclusions in pyrite.
2. Within quartz-carbonate stringers parallel or at an angle to the foliation. The gold mineralogy is more complex in this type as telluride minerals are more common than in the first type.

Gold minerals identified to date are native gold, calaverite and petzite. Altaite, chalcopyrite, sphalerite, galena, arsenopyrite, pyrite and chalcocite are also present. In general, native gold is finely disseminated and is interstitial to pyrite grains as well as occurring along grain boundaries and in cleavage planes of sheet silicate minerals.

Telluride minerals occur as free grains or associated with native gold or chalcopyrite or within pyrite.

Mining Methods

Presently the ore is being mined by open pit using loaders and 50 ton Euclid trucks. Planned stripping ratio in 1984 was 5 to 1.

Annual Tonnage Milled/Ore Grade

In 1984 ore mined and milled amounted to 634,000 tons with an average grade of 0.156 ounces per ton. Of this, 45% was milled on-site, 40% at the Est-Malartic mill and the remainder at Noranda's mill. Malartic and Noranda mills are 45 km. and 65 km. respectively, by road,from Doyon.

Gold Recovery

Gold recoveries were 95.86, 94.57 and 91.92% respectively at Doyon, Est-Malartic and Noranda Mills in 1984, with an overall average of 94.6%. The ore shipped to Est-Malartic is a 4 inch screen undersize product that is said to contain more pyrite and is somewhat more refractory to treatment than that milled on-site at Doyon.

Gold Production

Total production from all three mills in 1984 amounted to 93,457 ounces.

Milling Process

The process at Doyon is well desribed by D. Drouin in Reference No. 1. The conventional mill circuit consists of the following:

Three stage crushing using a jaw and two cone crushers.
Rod milling in open circuit in cyanide solution.
Ball milling with two-stage cyclone classification.
Thickening of ground product.
Primary leaching in 5 air-lift agitators.
Liquid/solids separation by vacuum drum filtration.
Secondary leaching in 6 air-lift agitators.
Liquid/solids separation by two-stage filtration.
Gold recovery by Merrill-Crowe process.
Solid waste pumped to tailings pond.

The process at Est-Malartic is basically similar to Doyon.

Operating Data

Refer to notes of meeting with George Hope in the Appendix.

Refractory Ore

Although the ore is not refractory, variations in the mill feed require prompt adjustments in the process to maintain satisfactory extraction. The presence of pyrite and chalcocite in the ore cause most of the problems. Chalcocite is said to occur in zones that make up about 1/3 of the mill feed.

Outlook

Operating costs per ounce of gold produced are presently quite low. Open pit reserves are adequate for 3 years operation. Although costs will probably increase when the mine goes underground the company appears to be confident that the operation will continue to be profitable for some years to come.

References

1. Proceedings of the 16th CMP Conference, 1984. Pp. 303-321.
2. CIM Special Volume 24, Pages 50-57.
3. Lac Minerals Ltd. Annual Report, 1984.
4. Can. Min. Journal, December 1984, P. 11.

KIENA GOLD MINES LIMITED

Location

Headframe and plant are located on Parker Island, Lac de Montigny, 10 km west of Val d'Or, in Northwestern Quebec.

Ownership

Falconbridge Ltd. holds 56.7% of the issued shares of Kiena.

Economic Metals

Gold

Ore Reserves

Reserves of proven and probable ore, as of December 31, 1984, including an allowance for mining dilution, were 5.9 million tons with a grade of 0.16 ounces per ton, sufficient for over 13 years of operation at current production levels.

Operating Costs

Expect costs to drop to C $275 - 280 per ounce this year with the new mill operating.

History

The Kiena mine came into production in late 1981. Until late 1984 the ore was custom-milled, mostly at Lamaque, but some went to other mills in the Val d'Or area. In August, 1983 Kiena commenced construction of their own mill at the minesite on Parker Island and this commenced operations late in September 1984 and by year-end was fully operational. Currently it is treating 30,000 tonnes per month.

Geology and Mineralogy

The main S-50 zone is a pyritic albite-chlorite breccia zone dipping at varying angles. Its average width is about 20 m but varies from less than one m to more than 40 m. The S-50 zone is contained within a thick envelope of altered peridotites which grade from a talc chlorite schist to a more massive peridotite with increasing distance from the ore zone.

Pyrite makes up 3 - 8% of the ore. Pyrrhotite hasn't been seen in the ore. There are very minor concentrations of chalcopyrite and sphalerite. No tellurides have been found in the ore.

Mining Methods

Kiena is reported to be the only mine in the Val d'Or area that uses completely trackless mining.

The mining scheme incorporates transverse rib pillars and secondary stub pillars for hangingwall support during the primary mining phase. After sections are mined and backfilled, the pillars are taken out providing 90 - 95% extraction of the ore.

Annual Tonnage Milled/Ore Grade

Ore milled in the full calendar year of 1984 amounted to 426,000 tons. Recovered grade was 0.156 oz/ton.

Gold Recovery

Recovery for 1984 was not published but the new mill is recovering 95% of the gold in the feed.

Gold Production

Gold production from the ore milled in 1984 was 66,657 ounces.

Milling Process

The new mill employs "state of the art" technology, including the following:

Semi-autogenous and ball-mill grinding in cyanide solution.
Thickening of ground product (cyclone overflow).
Gold recovery from thickener overflow by carbon in columns.
Cyanidation of thickener underflow (3 stages).
Gold adsorption from leach slurry by carbon-in-pulp (5 stages).

Stripping of carbon from static columns and carbon-in-pulp.
Recovery of gold in cathodes by electrolysis.
Smelting of cathodes in induction furnace to produce bullion.
Nitric acid washing and thermal regeneration of carbon.
Dewatering of leach tailings by disc filtration.
Destruction of cyanide in repulped tailings by sulphur dioxide and copper salt.
Two-stage cyclone classification to produce mine backfill.
Thickening of slimes to recover water.
Fines pumped to tailings pond.
Flotation of sand to scavenge gold, with concentrate recycled to grinding circuit.
Sands to mine for backfill.
Recycle of decant from tailings pond to the mill (future).

Operating Data

Refer to notes of plant visit in the Appendix.

Refractory Ores

No refractory ore reported.

Comments

The circuit for cyanide destruction and scavenger flotation for gold is still in the experimental stage.

Disc filtration of leach discharge is intended to recover cyanide and assist in minimizing the content of cyanide in scavenger flotation and in the sands recovered for mine backfill.

Outlook

With the on-site mill in operation, reasonable operating costs and generous ore reserves the outlook at Kiena for long-term operation is favorable.

References

1. Modern methods will boost productivity at operations of the new Kiena Gold Mine. Western Miner, January 1984, Pp. 15 - 18.
2. Advances in equipment design made Kiena's orebody economical. Can. Min. Journal, July 1984, Pp. 18 - 21.
3. Falconbridge Ltd. Annual Report, 1984.

LAC SHORTT GOLD MINE

Location

45 miles west of Chapais, Quebec.

Ownership

Corporation Falconbridge Copper 93%, Falconbridge Ltd. 7%.

Economic Metals

Mainly gold.

Ore Reserves, December 31, 1984

Proven and probable: 1.85 million tons @ 0.16 oz/ton.

Plus drill indicated possible: 0.62 million tons @ 0.14 oz/ton.

Operating Costs

Preproduction estimate C $230 /ounce for 2 years.

History

Property was owned by Opawica Explorations which maintains a 7.5% net proceeds interest in the mine.

A new mill, designed by Acres Davy McKee and Leslie Engineering went into operation in September, 1984. The mill capacity is 750 tons per day.

Geology and Mineralogy

Similar to the Bachelor Lake deposit in the area, the orebody at Lac Shortt is continuous, with the gold content per level fairly consistent and an average width of 20 feet.

The geology is complex. Gold is extremely fine and is present as free gold or as inclusions in pyrite (3-5% of ore).

Mining Methods

Mostly blasthole stoping with 50 m. stopes. Extraction will be 100% using cement and rock backfill. The orebody is continuous, permitting efficient mining.

Shaft has been developed to 613 m. level.

Annual Tonnage Milled/Ore Grade

For 1985 estimate 285,000 tons at 0.18-0.19 oz/ton Au.
Figures for 1984 were not published.

Gold Recovery %

Depends on milling rate but is approximately 95%.

Gold Production

Estimate 50,000 oz. in 1985.

Milling Process

Conventional crushing and grinding, using equipment moved from Falconbridge's Manibridge Mill. Cyanide leaching is followed by carbon-in-pulp gold extraction, stripping, electro-winning and smelting of cathodes.

The mill was designed for 750 tons per day but has been operating about 100 tons per day above this rate recently.

The ore is ground to at least 90% - 200 mesh.

Leach retention time is 40 hours.

Waste solids are pumped to a large, shallow pond where cyanide compounds are expected to degrade naturally.

Refractory Ore

The ore is not refractory to gold extraction.

Outlook

The orebody is open at depth and to the east, and the hoist is capable of handling 1,500 tons/day.

At the present price for gold, the operation will probably continue for several years at least.

It is expected that 50,000 ounces of gold will be produced in 1985 and then production will drop to 40,000 to 50,000 ounces per year for the 8-9-year life of the mill. Milling rate might be increased.

References

1. Lebel, P. Construction innovations up production at Lac Shortt. CMJ, May 1984, Pp. 23-27.

2. Jones, Allan. Is Lac Shortt triggering new gold mines in the area? Northern Miner, July 19, 1984, P. B-15.

3. Northern Miner, Oct. 4, 1984. Pp. A1, A14.

4. Personal communication, F. Pickard and S. Fekete.

5. Falconbridge Ltd., Annual Report, 1984.

LAMAQUE MINE

Location

The mine is adjacent to the south-east of the town of Val d'Or in Northwestern Quebec.

Ownership

Lamaque Mine is a division of Teck Corporation.

Economic Minerals

Gold with minor silver.

Ore Reserves

Reserves as of Sept. 30, 1984 were 425,000 tons at a grade of 0.148 oz./ton.

Operating Costs

For fiscal 1984,ending September 30th,operating costs averaged $54.48/ton of ore milled or C$478.23 per ounce of gold produced.

History

Production began at Lamaque in 1935 at a rate of 250 tons per day. The capacity of the mill was increased over the years to 2,100 tons per day in 1953, operating seven days per week. In 1973, due to a shortage of ore, the mill was reduced to 1,800 tons per day on a 5-day week. In recent years mine production has fallen off and the mill has treated ores on a custom basis to supplement Lamaque mine production. A 3 year contract to process ore from Kiena terminated in January, 1985. The Lamaque mine and mill were scheduled to close during March, 1985.

Geology and Mineralogy

The major portion of gold ore at Lamaque came from a zone centered around a chimney-like mass or plug of intrusive rock, consisting of light-colored granodiorite surrounded by a broad rim of darker hornblende diorite. Surrounding the plug is an assortment of typical greenstone volcanic rocks.

The principal vein constituents are quartz, tourmaline, and carbonate (ankerite, calcite or dolomite), with lesser quantities of pyrite, pyrrhotite, chalcopyrite, scheelite, fuchsite, gold and tellurides. Rarely found are sphalerite and galena.

Gold occurs in the native state as small visible specks, patches or veinlets in or very close to vein material. It also occurs in auriferous pyrite and, uncommonly, in tellurides.

Mining Methods

Seven shafts and two internal shafts were sunk to service the mine, with a bottom level 3,600 ft. below surface. Stoping methods included open stoping for flat-lying veins, shrinkage stoping for steeper veins, cut-and-fill for high-grade steep-dipping veins in incompetent zones, transverse cut-and-fill stoping for closely spaced veins and stringers and blasthole stoping for large, irregular-shaped, low-grade ore zones.

Annual Tonnage Milled/Ore Grade

For the fiscal year of 1984, ending September 30, 301,805 tons of ore averaging 0.113 oz./ton were milled. (Excludes custom ore).

Gold Recovery, %

Gold recovery in fiscal 1984 averaged 94.66%.

Gold Production

Production of gold for the same period was 32,414 oz.

Milling Process

The process consisted of conventional crushing and grinding, cyanidation of the total ore, liquid-solids separation by filtration, and gold recovery by zinc precipitation. There were six identical grinding circuits and three parallel leaching circuits.

A backfill plant returned 60% of the tailings to the mine.

Refractory Ores

No refractory ores have been reported.

Outlook

Limited ore reserves and high operating costs per ounce of gold suggest that the property will not reopen unless there is a sustained rise in the price of gold.

References

1. Milling Practice in Canada, CIM Spec. Vol. 16, 1978, Pp. 67, 68.
2. CIM Reporter, September 21, 1980, Pp. 3-5.
3. E & MJ December, 1981, Pp. 96, 97.
4. Teck Corporation Annual Report, 1984

MONTAUBAN MINE

Location

About 50 miles west of Quebec City.

Ownership

Muscocho Explorations Ltd. (84%), Soquem (16%).

Economic Metals

Gold and silver.

Ore Reserves

Established ore reserves in the zone being mined, early in 1985 were reported to be adequate for 3-4 years at 400 tonnes per day.

Operating Costs

To September 30, 1984 the year-to-date average mine and mill operating cost, exclusive of future development work, averaged C$313 per ounce of gold produced (approximately U.S.$239).

History

The property was mined intermittently for lead and zinc between 1911 and 1965. Muscocho began drilling for gold in 1975 and have since outlined two gold deposits known as the North and South zones.

The North Zone was opened up for mining by means of a main haulage ramp and development of three operating levels. Milling commenced in May, 1983 using a modular processing mill from the Camlaren operation in the N.W.T. and production has continued steadily since 1983.

Geology and Mineralogy

Geologically, Montauban is the only producing gold mine in Canada's immense Grenville Structural Province.

Gold occurs in one horizon in an area of metamorphased volcanic and sedimentary rock.

Mining Methods

The North Zone is being mined by means of a 1,920 foot sloping access ramp, using shrinkage stoping.

Annual Tonnage Milled/Ore Grade

In 1984, daily mill throughput averaged 273 tonnes per day in the first quarter, increasing to 335 tonnes in the second quarter and 363 tonnes in the third quarter. Average grade of ore treated in the third quarter was 4.90 grams per tonne (0.16 oz. per tonne). Early in 1985 the mill was running at an average rate of 380 tonnes/day and ore storage bins were being installed to increase the rate to more than 400 tonnes per day. Assuming an average 400 tonnes per day in 1985, throughput will be 146,000 tonnes for the year.

Gold Recovery %

Average recovery in the third quarter of 1984 was 91.3%.

Metal Production

Gold and silver production in 1984 totalled 17,071 ounces and 20,366 ounces respectively. Metal production in the third quarter of 1984 totalled 4,796 ounces of gold and 5,316 ounces of silver. If grade and recovery are comparable in 1985 and the milling rate averages 400 tonnes per day, gold production will be 21,100 ounces and silver production 23,400 ounces for the year.

Milling Process

The process is conventional, consisting of crushing and grinding, cyanide leaching and zinc precipitation. In January, 1984 a jig was added to the grinding circuit to catch some coarse gold not being recovered by cyanidation.

Refractory Ore

No refractory ore has been reported but gold values are very erratic, requiring careful blending to maintain a consistent grade of mill feed.

Outlook

The company has no long term debt and Muscocho's directors are looking confidently at the future.

References

1. Canadian Mining Journal, May, 1984. Pp. 31-35.
2. Northern Miner, Feb. 7, 1985, P. A12.
3. Muscocho Explorations Annual Report, 1983 and Interim Report. Sept. 30, 1984.
4. North American Gold Mining Industry News, March 15, 1985, Pp. 1, 36.
5. CIM Bull. March, 1976, P. 87.

SIGMA MINE

Location

Two km. east of Val d'Or in Northwestern Quebec.

Ownership

Dome Mines Ltd. has a 65% interest in Sigma Mines (Quebec) Ltd.

Economic Metals

Gold.

Ore Reserves

Proven extractable ore reserves, after allowing for dilution, at the end of 1983, were estimated to be 978,000 tons at an average grade of 0.196 ounces of gold per ton. In addition, there were 192,000 tons of broken ore.

At the Sigma No. 2 Mine (formerly Quebec Gold Belt Mines Ltd.) total probable reserves were estimated to be 602,000 tons with an average grade of 0.087 ounces per ton.

Operating Costs

For the first three quarters of 1984, operating costs were $45.90 per ton of ore milled or $390. per ounce of gold produced.

History

The Sigma mine has operated continuously since 1937. Initial milling rate was 300 tons per day. In 1941 the capacity was raised to 1,100 tons per day. The lowest level in the mine is 5,860 ft.

Production from the No. 2 mine, an open pit, began during the third quarter of 1984.

Geology and Mineralogy

Sigma Mines lies south of the Bourlamaque granodiorite batholith in a belt of volcanic rocks which has been invaded by various types of intrusives.

Gold mineralization is confined to quartz-tourmaline pyrite veins that occupy fractures and shear zones. Although small nuggets and blebs of free gold occur, most of the gold is closely associated with pyrite, occupying micro fractures within the crystals. Minor amounts of chalcopyrite, scheelite and, occasionally, tellurides occur in the ore zones.

Mining Methods

Various methods of mining are used including shrinkage stoping, room and pillar, cut-and-fill and other methods. The No. 2 mine is an open pit.

Annual Tonnage Milled/Ore Grade

For the first nine months of 1984, 353,400 tons of ore were milled with an average recovered grade of 0.118 ounces per ton.

Gold Recovery

Gold recovery has not been reported for 1984 but in the previous three years it averaged 96.4%.

Gold Production

For the first three quarters of 1984 gold production was 41,536 ounces.

Milling Process

The mill process and equipment appear to have changed little since 1941. The flowsheet includes the following:

Jaw and cone crushing.
Grinding in four stages, using 3 ball mills and a tube mill (operates in cyanide).
Thickening.
Cyanidation in 3 stages using Dorr agitators.
Liquid/solids separation by one stage of wash thickening and two stages of filtration.

Gold recovery by Merrill-Crowe process and smelting.

Two-stage classification to produce backfill (since 1957).

Fine waste pumped to tailings pond.

A process flowsheet, together with a description of mining and milling operations, are presented in the Appendix.

Refractory Ore

No refractory ore reported.

Comments

Sigma adds litharge in the grinding circuit to control ferrocyanide at the rate of 0.035 lb./ton.

Recently unsatisfactory levels of lead have appeared in the gold precipitate. Sigma are working with the Centre de Recherches Minerales on the problem.

Outlook

Sigma management believe that, with the open pit mine in operation, the mine life will be extended at least five years.

References

1. CIM Reporter, September 21, 1980, Pp. 5,6.
2. E & MJ, December, 1981, Pp. 92-94.
3. Sigma Mines (Quebec) Ltd. Annual Report, 1983 and Interim Report, September 30, 1984.
4. Telephone conversation, J.G. St. Jean, Mill Superintendent.

AGNICO-EAGLE SILVER DIVISION

Location

The Penn silver concentrator is located in Coleman township, 1½ miles southeast of the town of Cobalt in Northern Ontario. In 1983 about 75% of the mill feed came from the nearby Beaver - Temiskaming Mine, 17.5% from the Castle Mine near Gowganda and the balance from the Langis mine just north of New Liskeard.

The company also operates a silver refinery at Cobalt.

Ownership

Agnico-Eagle Mines Ltd. was formed by the amalgamation of Agnico Mines Ltd. and Eagle Gold Mines Ltd.

Economic Metals

Silver. Cobalt is concentrated but is not marketed.

Ore Reserves

Detailed reserve figures are not published but apparently are being maintained. The Beaver-Temiskaming mine is reported to have at least 3 years supply of good grade ore, more than they have ever had before.

Operating Costs

For nine months ending September 30, 1984, operating costs were $134.88 per ton milled or $5.28 per oz. of silver produced.

History

Agnico-Eagle Mines Ltd. is a consolidation of a group of mines and companies that commenced operation in the early years of this century. The company controls or leases a number of properties in the Cobalt - Gowganda area. The Penn concentrator commenced operations in July 1957.

Geology and Mineralogy

Most of the silver in the district is in the form of more or less pure native metal. The remainder is in the form of argentite or in a variety of complex silver minerals including arsenides of cobalt, nickel and other metals. Native silver often occurs with calcite. Host rocks are mainly sediments, diabase and lavas.

Mining Methods

Underground mining of narrow veins, probably using shrinkage stoping and/or cut and fill.

Annual Tonnage Milled/Ore Grade

For nine months ending September 30, 1994 the Penn Mill treated 43,310 tons of ore with an average recovered grade of 25.54 oz. of silver per ton. (Calculated feed grade at 97% recovery = 26.3 oz./ton).

Silver Recovery, %

For the full year of 1983 silver recovery averaged 96.2%.

Silver Production

For nine months ending September 30, 1984 silver produced in gravity and flotation concentrades amounted to 1,106,011 oz.

Milling Process

The 300 TPD concentrator flowsheet is unchanged from that reported in 1978. It consists of conventional crushing, ball milling to 45%-50% minus 200 mesh, jig concentration,table concentration followed by flotation of table tailings.

Silver recovery by jigging is 85-90%, by tabling 1½-2% and the balance by flotation.

Concentrates are shipped to the refinery in Cobalt. The flotation concentrate is reground and cyanided to recover silver.

Refractory Ore

The ore is not refractory to silver recovery. Cobalt is recovered in concentrates, associated with arsenic. Apparently it is not recovered in the refinery.

Outlook

The company has extensive experience in mining and milling silver ores and, assuming reasonable silver prices, will probably continue operations for several years.

Cobalt and other metals may ultimately be recovered from refinery residues.

References

1. Milling Practice in Canada, CIM Spec. Vol. 16, 1978, Pp. 50-52.
2. Agnico-Eagle Mines Ltd. Annual Report 1983 and Interim Report, September 30, 1984.
3. Telephone conversation with W. Montgomery, Mill Superintendent.

ASARCO'S AQUARIUS MINE

Location

The mine is located 22 miles east of Timmins, in Northern Ontario.

Ownership

The mine is being developed by Asarco Inc.

Economic Metals

Gold

Ore Reserves

Reserves are 205,000 tons with an average grade of 0.22 ounces per ton.

Operating Costs

Not available.

History

The deposit was discovered by Asarco. A shaft was sunk in 1982 and 8,000 tons of ore were processed in a test mill to compare recoveries with sampling results. A 300 TPD was built on the property and was operated for 8 months in 1984.

Geology and Mineralogy

Coarse gold occurs in quartz veins cutting a carbonate zone.

Mining Methods

Probably shrinkage stoping.

Annual Tonnage Milled/Ore Grade

In 1984 the mill processed 32,700 tons at a recovered grade of 0.215 ounces per ton during an 8 months test run to confirm the grade of ore and continuity of the veins.

Gold Recovery, %

At least 94%.

Gold Production

Gold produced in concentrates amounted to 7,030 ounces during 8 months operation in 1984.

Milling Process

The ore is crushed and gold recovered by jigging. The jig concentrates, containing 600-800 ounces per ton are upgraded by tabling to 2,000 ounces per ton and shipped to Asarco's smelter in Montana.

The mill also has a flotation circuit but it has been used to process custom ore from Kidd Creek. There is no cyanidation circuit.

Refractory Ore

The ore is not refractory.

Outlook

Asarco intends to develop the mine and go into regular production when the price of gold goes up.

References

1. Canadian Mines Handbook, 1983-84, P. 37.
2. Personal communication with R. S. Gray.

CAMPBELL RED LAKE MINE

Location

Campbell Red Lake Mines is located at the town of Balmertown in the Red Lake area of Northwestern Ontario.

Ownership

Dome Mines Ltd. owns 57% of Campbell's shares.

Economic Metals

Primarily gold. Crude arsenic dust is produced as a by-product.

Ore Reserves

Extractable reserves on December 31, 1984 after allowing for dilution were 2,026,000 tons at an average grade of 0.622 ounces per ton. This does not include 265,000 tons of broken ore.

Operating Costs

Operating costs in 1984 were $75.78 per ton of ore mined, equivalent to $140 per ounce of gold produced.

History

Milling commenced in 1949 at a rate of 350 TPD. The mill has been expanded over the years, including a 30% expansion in 1981 which increased the milling capacity in 1982 to an average 1,072 tons per calendar day.

When the mill first went into operation it was shown that maximum gold recovery by straight cyanidation would not exceed 80%. To improve the recovery, a single-stage roasting plant went into operation in August, 1949 and, by cyaniding the roaster calcine, gold recovery was boosted by about 10%.

In February, 1956 a second-stage roaster was added to permit better control of bed temperature and minimize clinkering problems caused by the presence of antimony. (Antimony in the concentrate in recent years has been low.)

Until 1974 off-gas from the roasters was discharged untreated through a 125-foot stack. In 1974 a new gas cleaning plant, consisting essentially of electrostatic precipitators and baghouses, went into operation.

Campbell's roasting plant has performed very well. They modelled their plant on Giant Yellowknife's and benefited from Giant's experience.

Geology and Mineralogy

Host rocks are andesite and diorite. Gangue minerals are mainly quartz, with carbonates and talc.

Gold occurs as coarse free gold and as gold closely associated with sulphides and particularly arsenopyrite. The major sulphides are pyrite, pyrrhotite, arsenopyrite, stibnite, sphalerite and chalcopyrite in order of abundance. Pyrrhotite, arsenopyrite and stibnite are the refractory minerals that complicate the metallurgical process.

Generally the mineralization is extremely fine and there is intimate association between sulphides and quartz.

As mining extends to deeper levels the proportion of sulphides is decreasing.

Mill head assays in 1984 averaged as follows:

Au, oz/ton	0.63
S, %	1.09
As, % (calc'd)	0.41
Fe, % (calc'd)	7.60

Mining Methods

In 1984, 62% of the ore came from cut-and-fill, 30% from shrinkage and 8% from development.

As a result of rock bursting in 1982 and 1983, stoping in one area has been halted. Efforts continue to develop safe mining methods at deep levels. A change to cut-and-fill mining has been accelerated.

Annual Tonnage Milled/Ore Grade

In 1984, 395,000 tons of ore were milled, with an average grade of 0.573 ounces per ton.

Gold Recovery, %

In 1984 total extraction was 94.5%, distributed as follows:

Gravity Concentration	
Bullion from Jigs	38.3%
Bullion from Calcine	13.2%
Cyanidation	
Bullion from Precipitate	43.0%
(includes 5.5% extracted from flotation tailings)	

Total extraction includes 496 ounces (0.2%) recovered from tailings water treated in a carbon plant.

Tailings losses at the mill were distributed as follows:

Cyanided flotation tailings	3.9%
Cyanided calcine	1.2%
Tailings solution	0.6%

Gold and Arsenic Production

Gold production in 1984 amounted to 213,946 ounces.

Arsenic dust shipped in 1984 was 1,371 tons with an average grade of 93.35% $As_2 O_3$.

Milling Process

Campbell's process is fairly complex, due to the necessity of floating and roasting sulphides in the ore in order to extract gold locked in the sulphides.

A flowsheet of the process is presented in the Appendix.

Essentially the process consists of the following:

Conventional crushing and grinding circuits.

Coarse gold recovery from ground product by jigging and tabling.

Flotation to separate sulphides from gangue.

Cyanidation of flotation tailings, followed by filtration.

Classification of tailings residue with sand going to mine backfill and fines to tailings pond.

Scavenging gold from flotation concentrate on blankets.

Two-stage fluid bed roasting of concentrates.

Gas cleaning by electrostatic precipitation, followed by cooling and bag filtration to recover arsenic trioxide.

Waste gases vented up 125-foot stack.

Gravity concentration of quenched roaster calcine to recover free gold.

Regrinding of calcine.

Cyanidation of reground calcine and ESP dust.

Filtration of leached calcine and dust.

Gold in solution from both leaching circuits recovered by Merrill-Crowe system.

Gravity concentrates smelted to produce bullion.

Operating Data

Refer to notes of visit to Campbell, in the Appendix.

Types of Refractory Ore

Refractory ore contains gold which is so intimately associated with arsenopyrite that it requires roasting or chemical dissolution to release it.

Percent of Gold in Refractory Ore

Approximately 15% of the gold in the total ore would not be recovered by straight cyanidation if the concentrate wasn't roasted.

Percent of Refractory Ore to Total Ore

Possibly some zones contain ore that is not refractory.

Current Status of Treatment of Refractory Ores

By floating sulphides, roasting the concentrate and treating the calcine by gravity concentration and cyanidation almost all the refractory gold is recovered.

Sulphur level in the ore has been dropping at lower mining levels, making it difficult to maintain the minimum sulphur in concentrate to maintain an autogenous roast. Campbell may have to add elemental sulphur to the concentrate.

Problems Related to Presence of Refractory Ores

Problems are mainly confined to the roasting process.

Sulphur dioxide is not removed from the off-gas and, whereas emissions apparently meet regulations, it may cause some damage to the environment.

Campbell have a well-designed and maintained roaster operation and it recently added an air-conditioned control room where the operators control the whole operation. It is becoming increasingly difficult to meet stringent in-plant regulations for arsenic and to find operators who are willing to go into the plant for clean-up and maintenance.

At times of depressed markets for arsenic, storage of the dust can be a problem.

References

1. CIM Congress Volume, 1957. Pp. 124, 125.
2. Milling Practice in Canada, CIM Spec. Vol. 16, 1978, Pp. 54-56.
3. Wright, K.P. "Fluid Bed Roasting Practice in the Red Lake Camp," CIM Bull. Aug., 1961. Pp 595-600.
4. Roberts, J. Scott. "Controlling roaster off-gases at Campbell Lake Mines," CMJ, July 1976, Pp. 54-56.

DETOUR LAKE MINE

Location

The mine is located 125 miles north-east of Timmins, in Northern Ontario, just 8 miles west of the Quebec border.

Ownership

The Detour Lake Mine is a 50/50 joint venture between Campbell Red Lake Mines and Amoco Canada Petroleum Co. Campbell is the operator of the mine.

Economic Metals

Gold. There is a small amount of copper in the ore which might be profitable to mine if the price of copper rises substantially.

Ore Reserves

Total proven and probable ore reserves in place, after allowing for dilution, as of December 31, 1984, were 10.9 million tons at an average grade of 0.128 ounces per ton.

Operating Costs

Mine operating costs, including depreciation averaged $53.23 per ton milled in 1984, or $591 (US$ 455) per ounce. The cash cost per ounce was $447 (US $345).

History

Milling commenced in July, 1983 with ore from an open pit. By March, 1984 the 2,750 TPD mill was operating at only 2,550 TPD. Gold production was less than anticipated due to dilution in the open pit, harder ore than expected and excessive wear on the liners in the semi-autogenous grinding mill. For the full year 1984 gold production was 21% below forecast.

It is planned to go into underground operation in 1987. At year-end 1984 a headframe and hoist had been completed and shaft sinking had commenced.

Geology and Mineralogy

The gold mineralization is concentrated within and adjacent to a cherty-tuff horizon. The older footwall ultramafic rocks are generally altered to talc-chlorite, talc-carbonate or tremolite-actinolite. Basaltic material overlies the cherty-tuff. Gold values are found in all three rock units and are generally related to both the degree of quartz fracturing and the amount of sulphide minerals present, especially chalcopyrite.

Many mineral species constitute the veins, including quartz (most common), carbonates, actinolite-tremolite, chlorite, epidote, biotite and feldspar.

Sulphide mineralization, primarily pyrite, pyrrhotite and chalcopyrite constitute a large volume of the veins and extend as well into the wall rocks in diminishing quantity.

Gold is very fine grained, rarely visible and shows a distinct association with chalcopyrite and possibly pyrite but not with pyrrhotite. Gold-bismuth tellurides have been observed in the quartz veins.

Mining Methods

Currently all production is from an open pit.

Annual Tonnage Milled/Ore Grade

During the year 1984, 876,000 tons of ore with an average grade of 0.097 ounces of gold per ton were milled.

Gold Recovery, %

Gold recovery in 1984 was 93%.

Gold Production

Production for the year was 78,958 ounces.

Milling Process

The Detour mill utilizes "state of the art" technology including semi-autogenous grinding and carbon-in-pulp extraction of gold. The major unit operations of the process are as follows:

Jaw crushing to 8 in.
Semi-autogenous grinding to 74%-200 mesh.
Pre-aeration with lime.
Thickening.
Leaching with cyanide, lime and lead nitrate.
Gold adsorption by carbon-in-pulp.
Pressure stripping of gold from carbon.
Electrolytic deposition of gold, followed by smelting to produce bullion.
Alkaline chlorination of waste slurry.
Flotation of copper (not operating).
Solid wastes pumped to tailings pond.

When the mill was designed it was intended to raise the capacity to 4,000 TPD, possibly in 1987, and, at that time a pebble mill would be added in line with the existing 5.2m by 8.5m Allis-Chalmers semi-autogenous mill. At the present time grinding is done in one stage in the semi-autogenous mill.

Operating Data

Current reagent additions, excluding alkaline chlorination are as follows:

Cyanide	1.51 Kg/t
Lime	2.48 Kg/t
Lead nitrate	0.07 Kg/t

Lead nitrate is added solely to the cyanide leaching circuit. The plant has operated without it but the operators feel that it is necessary.

Types of Refractory Ore

The ore is not very refractory.

Problems in Ore Treatment

Due to the presence of iron sulphides, particularly pyrrhotite, the ore is preaerated prior to leaching. Despite this treatment, cyanide consumption is high and relatively high concentrations of cyanide complexes are formed in leaching. Not all of these are destroyed by alkaline chlorination.

Outlook

The joint partners were expected to meet in mid-April to decide whether to close the mine due to the high operating costs per ounce of gold.

References

1. Knoll, Kerry. "Beating the clock and budget at Canada's largest gold mine." CMJ May, 1984, Pp. 63-69.

2. Northern Miner, June 7, 1984, P. 24; Jan. 31, 1985, P. 19; March 28, 1985, Pp. 1, 6.

3. Campbell Red Lake Mines Ltd., Annual Report, 1984.

4. Personal communication with Ken Wright.

ARTHUR W. WHITE MINE (DICKENSON)

Location

Balmer Township, Red Lake Area of Northwestern Ontario.

Ownership

Dickenson Mines Ltd., 65%, Sullivan Resources Ltd.,35%.

Economic Metals

Gold. Minor silver.

Ore Reserves

Reserves as of Dec. 31, 1984 were as follows:

	Short Tons	Grade Ounces Gold Per Ton
Proven	1,240,000	0.332
Probable	1,233,000	0.317
Possible	810,000	0.279
Total	3,283,000	0.314

Operating Costs

Operating cost per ounce in 1984, including corporate costs, was CS310.

History

Milling commenced in December, 1948 using a 150 TPD mill from the Gold Eagle Mine on McKenzie Island. Capacity was increased to 470 TPD by 1959 and to 700 TPD by mid-1983. The original mill used straight cyanidation and recoveries were 75 to 80%. In 1951 the circuit was modified to include jig concentration and amalgamation of the concentrate.

At the same time a flotation circuit was installed to recover sulphides before cyanidation and a single-stage fluid bed roaster was put in to roast the concentrate. Roaster calcine and flotation tailing were both cyanided. In 1954, the quantity of concentrate began to exceed the roaster capacity. A major change in the circuit was made with cyanidation of the total mill feed. The flotation circuit was relocated to the tail end of the circuit to recover only the sulphide-locked gold.

A backfill plant was added in 1959.

In 1979 the roasting plant was shut down due to environmental problems. It has since been demolished.

In 1981 a series of intensive aeration tanks were installed to break down arsenopyrite and release the locked gold. This circuit did not meet expectations and the tanks were subsequently removed from the plant.

Since 1981 the mill has been expanded to its present capacity of 800 tons/day, largely through additions in the grinding circuit and in the leach agitators. With moderate additional expense, largely for a new fine ore bin, the capacity could be raised to 1000 tons/day.

Geology and Mineralogy

The Dickenson Mine is underlain by volcanic and sedimentary rocks of Pre-Cambrian age. Gold is found in the volcanics. It occurs in three specific types of alteration environments:

1. Massive iron-dolomite veins. Gold occurs in cross-cutting quartz veins and fine-grained breccias within the iron-dolomite veins.
2. Small patches of moderate to intense silicification. These patches are characterized by a pervasive replacement of the rock by quartz. Gold occurs as disseminations within the silica-replaced rock and as high grade pockets in the quartz veins.

3. Sulphide disseminations. This environment consists of rocks containing anomalous concentrations of disseminated secondary sulphides, dominated by pyrite, pyrrhotite and arsenopyrite. Sulphide abundances are highly variable but average about 5% combined. Gold occurs in intimate association with the disseminated sulphides.

In places the ore also contains small amounts of sphalerite, magnetite, stibnite, chalcopyrite and pentlandite. Pyrrhotite content is very low.

Overall, gold occurs in the free form as relatively large particles, attached to or enclosed in sulphides as fine gold, or as sub-micron particles in sulphides, particularly in arsenopyrite. However, arsenopyrite doesn't always contain gold.

Mining Methods

Mostly cut and fill. Blasthole methods introduced in the early 1980's resulted in excessive dilution and cut and fill was reintroduced.

Annual Tonnage Milled/Ore Grade

In 1984, 134,000 tons of ore with an average grade of 0.340 ounces per ton were milled.

Gold Recovery %

Amalgamation	15 - 17%
Amalgamation plus cyanidation	83.8%

An additional 2.4% of the gold was recovered in concentrate which was shipped to Boliden in Sweden.

Gold Production

Production in 1984 amounted to 38,300 ounces in bullion form and 1,200 ounces in concentrate.

Milling Process

Surface crushing, 3 stages to 0.5 - 0.75 in.

Grinding, 3 stages to 70% - 200 mesh in cyanide.

Jig concentration and amalgamation.

Cyanide leaching, single stage.

Vacuum drum filtration, single stage.

Gold recovered by Merrill-Crowe process.

Cyanidation tailings floated to recover concentrate containing locked gold.

Tailings cycloned twice to produce mine backfill.

Presently adding scavenger circuit using carbon adsorption.

It is planned to discontinue amalgamation next year and replace it with a gravity concentration circuit similar to Campbells'.

Operating Data

Leaching pH 10.5 - 11.0 (don't titrate for lime)

Calcium cyanide added in rod mill and 3 other points.

Cyanide level (as NACN) Front end 1.5 lb./ton of solution.

Precip. 1.0 lb./ton of solution.

Ret. time - 96 hr. @ 600 TPD.

Mill uses calcium cyanide, which is said to be 16% less costly than sodium cyanide, and it also contains some lime.

Reagent Consumptions:

Cyanidation - 2.6 lb./ton cyanide (NACN equiv.)
- 1.5 lb./ton lime (CAO)
- 0.5 lb./ton lead nitrate

Flotation - 0.062 lb./ton SIBX
- 0.29 lb./ton copper sulphate
- 0.14 lb./ton Dowfroth 250
- 0.028 lb./ton Pennfloat P-3
- 0.044 lb./ton sodium sulphide

(Copper sulphate neutralizes excess cyanide and activates sulphide minerals.)

Cyanamid reagents for free gold have been tested but were not unusually effective.

Percent of Gold in Refractory Ores

Recovery by cyanidation testing on various ore zones varies from 68% to 95%, however, the proportion of ore that is refractory has not been calculated.

Types of Refractory Ore

Mostly ore containing arsenopyrite with part of the gold in submicron form.

Current Status of Treatment of Refractory Ores

Roasting is no longer practiced; the plant has been demolished.

Dickenson is currently producing a sulphide concentrate which is shipped to Boliden in Sweden on a 5-year contract. Due to shipping and treatment costs, emphasis is placed on concentrate grade, rather than recovery. Gold recovery is about 5% of the mill feed in a concentrate assaying about 1 ounce per ton.

At the present price of gold there is little profit being made on this concentrate.

(Dickenson's Annual Report indicates only 2.4% of the gold in concentrate. The flotation plant apparently operated for only part of the year.)

Since it is barely profitable to ship concentrate to Sweden, Dickenson would ultimately like to treat its own concentrate.

Dickenson believes that it could scavenge 10% of the gold in a lower grade concentrate. The company is investigating methods of treating this concentrate. Best results have been achieved by oxidative pressure leaching (98% extraction) but the cost of oxygen is prohibitive. Ontario Research Foundation's process using Caro's acid extracted only 37% of the gold. B.C. Research tested a combination of atmospheric oxidation and bacterial leaching, with filtration between these stages; best results indicated 85.9% extraction.

Prochem has a method of treatment which includes atmospheric oxidation with an oxidant (Reagent "X") and this may hold some promise.

It may be possible to separate pyrite, which carries little gold, from the arsenopyrite, thereby reducing concentrate treatment costs.

Problems Related to Presence of Refractory Ores

With roasting no longer practiced there are no particular problems related to the presence of refractory ores.

Comments

Dickenson didn't use the two-stage roasting process pioneered by Giant and Campbell. Its single-stage process never operated entirely satisfactorily.

Improvements in flotation separation of arsenopyrite from pyrite, and in treatment of arsenical concentrates, could result in Dickenson treating their arsenical concentrate on site.

References

1. Andrews, A.J. Alteration, Metamorphism and Structure Associated with Archean Volcanic - Hosted Gold Deposits, Red Lake District. Ont. Geol. Surv. Misc. Paper 119, pp. 162 - 166.
2. The Staff, 1974: Dickenson Mines Limited, 36 pp, unpublished.
3. Northern Miner, Jan. 17, 1985, pp 1, 24.
4. The Mill: Dickenson Staff, July, 1983, unpublished (see Appendix)
5. Scales, Marilyn. Dickenson Expansion Reduces Milling Costs, CMJ, June, 1983, pp 30 - 34.
6. Dickenson Cashes in on New Mining Strategy, CMJ, June, 1983, pp 24 - 28.
7. Visit to Dickenson Mine, February 21, 1984.
8. Dickenson Mines Ltd. - Annual Report, 1984.

DOME MINES LIMITED

Location

The Dome mine is located 3 Km southwest of South Porcupine in the Timmins area of Northern Ontario.

Ownership

Dome Petroleum Ltd. is the major shareholder in Dome Mines with 39.3% of its shares.

Economic Metals

Gold.

Ore Reserves

As of December 31, 1984 proven extractable reserves, including dilution were estimated to be 2,600,000 tons at an average grade of 0.170 ounces per ton. Broken ore is included.

Operating Costs

In 1984 operating costs at Dome averaged $49.30 per ton or $358 (US $276) per ounce of gold produced.

History

Dome has operated continuously since 1912 except for 1½ years during World War I and approximately one year following a fire in 1929. Over the years many improvements were made in the process but cyanidation has always been employed. Except for a short period, some method of recovering coarse gold has been practiced.

In 1981 a major program was started to expand output from 2,000 to 3,000 tons per day, and this program was completed in 1984. It included a new No. 8 shaft to a depth of 5552, new grinding facilities and upgrading and expansion of other process areas. A new tailings disposal area was completed in 1983.

Geology and Mineralogy

Gold occurs primarily as coarse native metal in quartz or ankerite veins. Host rocks are intermediate greenstone, conglomerate, slate and porphyry. Several tellurides, namely altaite, petzite and tellurobismuthite, have been found in the mine but constitute a very minor source of total gold production. Silver is recovered in the ratio of 1 part silver to 6 parts gold.

Sulphides average about 2 to 3% of the ore. Although pyrite and pyrrhotite are the dominant sulphides, chalcopyrite, sphalerite and galena are found locally in most ore types and are quite good indicators of gold content.

Mining Methods

In 1984 36% of the ore was extracted by cut-and-fill panel mining, 57% by longhole stoping and the remainder from development.

Annual Tonnage Milled/Ore Grade

Ore milled in 1984 amounted to 860,000 tons with an average grade of 0.142 ounces per ton.

Gold Recovery, %

Gold recovery improved to 97% in 1984 from 95.7% in 1983.

Gold Production

In 1984, 118,472 ounces of gold were produced.

Milling Process

The total ore is cyanided except for gravity concentrates which are smelted directly. Gold is extracted from leach solution by the Merrill-Crowe process.

Essentially the process consists of the following:

Primary crushing underground.

Two-stage cone crushing on surface.

Grinding by open circuit rod milling and closed-circuit ball milling (2-stage cyclone classification).

Jig concentrates recovered from primary cyclone underflow.

Jig concentrates upgrade by two-stage tabling and final concentrate smelted to produce bullion.

Thickening of finely ground ore.

Pre-aeration of thickener underflow.

Disc filtration.

Cyanide leaching with lime, cyanide and litharge.

Liquid/solids separation by single-stage drum filtration.

Gold precipitation with zinc, filtration and smelting.

Two-stage classification of tailings to produce mine backfill.

Fine tailings pumped to waste area.

Natural degredation of cyanide in tailings pond.

About 55% of the gold produced comes from the gravity concentrate.

Operating Data

Refer to Report to CANMET on Visit to Dome Mines Limited, October 16, 1984 by I. C. Edwards.

Refractory Ore

No refractory ore has been reported at Dome.

Comments

Dome's new tabling plant for jig concentrates, followed by smelting to produce bullion, replaces the old amalgamation. It is working well but may require more labor than the system used at Campbell Red Lake.

Pre-aeration is necessary at Dome because of the pyrrhotite in the ore. By filtering the pre-aerated slurry prior to leaching, some of the soluble salts generated by pre-aeration which may be detrimental in leaching, are removed. Filtration allows some barren solution to be recycled, thereby conserving the cyanide and gold in this barren.

Some cannon mixers have been installed at Dome and are reported to be working satisfactorily. Dome has also been testing the Joy Tailspinner as a means of improving the yield of mine backfill, and possibly the quality of the backfill.

Outlook

Dome is confident that the mine will continue to operate for many years to come. After 70 years operation over 50% of mine production came from above the 2,000-foot level and the deepest production level is only 4,000 feet.

References

1. Pabst, Marilyn. Rising gold price spurs new mine techniques, mill update at Dome. CMJ Dec. 1979, Pp. 24-26.

2. Pabst, Marilyn. Dome spends $91.2 million on expansion. CMJ, Oct. 1981, Pp. 29-37.

3. Dome expansion cuts costs increases rates to 3,000 TPD CMJ, May, 1984, Pp. 16-22.

4. CIM Reporter, Sept. 21, 1980, P. 7.

5. Colquohoun, R. H. Expansion of the Crushing and Milling Facilities of Dome Mines Limited, Proc. of the 16th Annual Meeting of the Canadian Mineral Processors, 1984, Pp. 322-343.

6. Dome Mines Limited, Annual Reports, 1983, 1984.

GOLDLUND MINE

Location

Goldlund is located 26 miles south-west of Sioux Lookout in Northwestern Ontario.

Ownership

Previously independent, the mine was taken over by Campbell Resources Inc. in March, 1984, through their subsidiary, Royex Gold Mining Corp. which holds 32% of Goldlund's shares.

Economic Metals

Gold

Ore Reserves

Published estimated reserves range from 200,000 tons at an estimated grade of 0.18 ounces per ton to 560,000 tons averaging 0.19 ounces per ton.

Operating Costs

Costs are not available but Royex reported that its share of Goldlund's losses in the fiscal year ending September 30, 1984 was approximately $400,000.

History

Regular production commenced in January, 1983 at a nominal rate of 200 tons per day, mining ore from underground and from an open pit.

Initially a concentrate was produced and shipped to Pamour's Schumacher mill for processing. In 1984 a mill expansion was under way to increase production to 350 tons per day and possibly to add a cyanide circuit. In the first quarter of 1985 the mine was shut down due to the dropping gold price.

Geology and Mineralogy

No information.

Gold Production

Royex reported that, during the fiscal year ending September 30, 1984 the mill processed an average of 223 tons of ore per day, averaging 0.14 ounces of gold per ton, and produced 7,300 ounces of gold.

Gold Recovery, %

Campbell Resources reported that gold recovery was 88% in the year ending June 30, 1984.

Milling Process

Conventional crushing, probably with jigging and flotation. A cyanide circuit may have been added in 1984.

Outlook

It would probably take a substantial increase in the price of gold to reopen the mine.

References

1. Canadian Mines Handbook, 1984-85, P. 163.
2. Campbell Resources Inc. Annual Report, 1984.
3. Joint Management Information Circular, Shareholders Meetings of Royex Gold Mining Corp. and Pezamerica Resources Corp., January 18, 1985.
4. Northern Miner, March 14, 1985, P. 11.

KERR-ADDISON MINE

Location

Kerr-Addison's mine and mill are located at Virginiatown, about 60 km east of Kirkland Lake in Northern Ontario.

Ownership

Noranda Mines Ltd. holds directly or indirectly 49.9% of the shares and Brenda Mines Ltd. holds 16.9%.

Economic Metals

Gold, minor silver.

Ore Reserves

At year-end 1984, mineable ore reserves, with allowance for dilution, in the proven and probable categories, were 607,608 tons at an average grade of 0.109 ounces of gold per ton, down from 927,903 tons at an average grade of 0.121 ounces per ton a year earlier.

Reserves are considered to be in the marginal category.

Operating Costs

Kerr-Addison doesn't report detailed cost information but the mine operating profit for 1984 was only $0.3 million compared with $5.1 million in 1983. These figures were before depreciation and amortization.

History

Milling operations at Kerr started in 1938 at a rate of 500 tons per day. The operation expanded in subsequent years to 4,500 tons daily by 1948. Flotation and roasting plants were added in 1955. Since 1960, with increasing mining depth, production rates have fallen off to 1,000 tons per day or less. Some custom ore has been processed in recent years.

Geology and Mineralogy

All the gold deposits are associated with the Larder Lake "break" which is located on a major contact between volcanics and sediments. The gold zones are classified locally as "green carbonate" ore and "flow" ore. There are indications that the "green carbonate" zones are replaced and altered basic volcanics. The "flow" zone appears to be a dacitic to rhyolitic flow which contains abundant pyrite. It is postulated that in the "flow" ore, gold was deposited contemporaneously with the pyrite and other rock-forming minerals.

In the carbonate zone gold occurs in the fine native form in stock works of quartz stringers. This zone contains less than 1% pyrite with traces of other sulphides.

The flow ore occurs in fractured vesicular lavas mineralized with gold-bearing quartz and pyrite. It contains 5 - 10% pyrite and traces of other sulphides. This ore is harder and more refractory than the other type. This is due to the fact that some of the gold particles in the pyrite are less than 10 microns in diameter.

Some graphite occurs in the mine and part of it is an active precipitator of gold.

Mining Methods

The mine is served by two shafts. No. 3 shaft extends to the 4,000 ft. level. No. 4 is an internal shaft, 1/2 mile north of No. 3; it extends from the 3,700 ft. level to 6,000 ft. level. A 36-inch conveyor carries ore from the 4,000 ft. level of the No. 4 shaft to the No. 3 shaft.

Tonnage mined by mining method during 1984 was 18% from square set stopes, 27% from cut and fill stopes and 52% from shrinkage and blasthole stopes.

Annual Tonnage Milled/Ore Grade

The Kerr Mine produced 378,530 tons in 1984 at an average daily rate of 1,037 tons and an average grade of 0.114 ounces of gold per ton. An additional 3,790 tons were toll milled for a customer.

Gold Recovery %

Gold recovery for 1984 averaged 97.22%. This includes an estimated 3 to 5% extracted from the reground pyrite.

Gold Production

Precious metal production from the Kerr mine in 1984 amounted to 42,063 ounces of gold and 1,758 ounces of silver.

Milling Process

Gold is extracted from the ore by cyanidation of the total ore followed by flotation of pyrite from the sand portion of the tailings and cyanidation of the reground pyrite concentrate.

In total the circuit consists of the following:

Conventional crushing and three-stage grinding in cyanide to 82% - 200 mesh.

Thickening, with gold recovery from the overflow by Merrill-Crowe process.

Cyanide leaching in air-lift agitators.

Liquid/solid separation by 2-stage CCD and one filtration.

Leach residues repulped and cycloned twice for backfill.

Secondary cyclone sands floated to recover pyrite.

Pyrite concentrate reground in batches and recycled to the leach circuit.

Flotation tailings to mine for backfill.

Slimes to tailings pond.

For many years the pyrite concentrate was roasted in a fluid bed reactor and the calcine was cyanided. Recently it was necessary to shut the roaster down due to an in-plant problem with arsenic. Early in 1985 it was decided to regrind the concentrate and return the ground product to leaching. Pyrite concentrate amounts to 15 tons per day. The concentrate is ground in 5-ton batches for 6 hours in an 8 ft. x 15 ft. ball mill with 1½ in. balls. The product is said to be 99.9% - 200 mesh.

This new system is working well and gold recovery is being maintained. It requires little operator attention compared with one operator full time each shift on the roaster. The heat previously generated in the waste heat boiler on the roaster is no longer required.

The graphite content of the ore is higher now than ever. Gold precipitation due to active graphite is prevented by adding kerosene to the grinding circuit.

It is planned to put in a pond for barren solution later this year.

Types of Refractory Ore

The carbonate ore was reported to be relatively soft and gave an extraction of 99% by straight cyanidation when ground to 90% - 200 mesh (Ref. 1, 1957). The flow type was reported to be harder and more refractory, yielding an extraction of only 90 to 95% at 90% - 200 mesh. The refractory nature of the ore was due to finely divided gold in pyrite. (Considering these extractions neither ore could be considered to be very refractory.)

Percent of Refractory Ore to Total Ore

In 1984 the carbonate ores provided 66% of the tonnage mined at a grade of 0.076 ounces per ton. The "refractory" flow zone ore made up the balance (34%) with a grade of 0.173 ounces per ton. As the better grade flow ores are depleted, the proportion of lower grade carbonate tonnage (non-refractory ore) will continue to increase.

Percent of Gold in Refractory Ore

From the above figures, 54% of the gold mined in 1984 was in the "refractory" flow ore.

Problems Related to Refractory Ores

No serious problems have been reported at Kerr from the treatment of refractory ore. The discharge of sulphur dioxide from the roaster stack, while undesirable, apparently met government guidelines.

There has been some problem or concern about the arsenic level in the plant due to the roasting process and as a result the roaster has been shut down.

Comments

Kerr-Addison was one of the first mills in Canada to recover backfill from tailings wastes and to use fluid bed roasting of sulphides.

Outlook

Unless there is a considerable increase in the price of gold or an exceptional discovery is made on the property the mill will likely close within the next few years.

References

1. The Milling of Canadian Ores, CIM Congress Vol., 1957. Pp. 97 - 113.
2. Milling Practice in Canada, CIM Vol. 16, Pp. 65 - 67.
3. CIM Reporter, Sept. 21, 1980, P. 8.
4. The Kerr-Addison Press, Jan. 1, 1981.
5. Kerr-Addison Mines Ltd., Annual Report, 1984.
6. Meeting with E.R. Ramsay, Mill Superintendent, March 25, 1984.

MACASSA AND LAKE SHORE MINES

Ownership

Macassa is a division of Lac Minerals Ltd. Lake Shore Mines Ltd. largest shareholder is Lac Minerals Ltd. (45.7%). In turn Lake Shore controls 53% of Lac Minerals.

Location

The Macassa mine and mill are located on the western outskirts of Kirkland Lake in Northern Ontario. The Lake Shore shaft is 4 km. east of Macassa.

Economic Metals

Gold, minor silver.

Ore Reserves

Proven ore reserves at year end 1984 at Macassa amounted to 1,067,000 tons with an average grade of 0.525 ounces per ton. Probable reserves were 377,000 tons grading 0.442 ounces per ton. Figures include allowances for dilution.

Proven reserves in the Lake Shore crown pillar on Dec. 31, 1984 were 77,000 tons with a grade of 0.370 ounces per ton. Other proven reserves at Lake Shore amounted to 312,000 tons with an average grade of 0.360 ounces per ton.

Operating Costs

Lac Minerals does not publish detailed cost data for individual operations but cost of gold produced at Macassa was reported to be $320 per ounce in 1984.

History

Milling commenced at Macassa in 1933 at a rate of 150 tons per day. This was increased to 400 tons per day in 1938 by the addition of a second grinding unit. Aside from improved classification and the addition of a tower mill to the circuit the mill has not been modified to any extent since 1938. Macassa is the longest operating mine in the Kirkland Lake area.

Today, Lac is mining Macassa from the original shaft system between the 5,000 foot and 6,400 foot levels, about one mile west of the headframe. To provide more economical access to present workings and to develop more reserves, Lac began sinking No. 3 shaft one mile west of the old shaft during 1983. The new shaft is expected to reach its target depth of 7,275 feet in late 1985.

The Lake Shore Mine, one of the richest gold mines in Northern Ontario, was closed in 1965 due to difficult ground conditions. In 1983 Lac began extracting ore from the main pillar of Lake Shore's shaft and most of the ore has been treated at Macassa.

Since the spring of 1984 Lac has been exploring old working areas in the vicinity of Lake Shore's No. 5 shaft and has built a new headframe and related facilities on this property. It is planned to remove old pillars and pockets of ore from Lake Shore and possibly develop new areas of gold mineralization.

Geology and Mineralogy

The gold-bearing ores at Macassa are associated with pre-ore faulting and fracturing. They consist chiefly of quartz veins, carbonated, silicified and pyritized syenite, porphyry syenite and augite syenite, which are enclosed in fissure-type veins, stockworks and vein breccias. Some molybdenum and lead tellurides occur in places. Some gold is completely encased in pyrite and for some time Lake Shore floated pyrite from cyanidation tailings from Lake Shore, Macassa and Wright-Hargreaves mills. The concentrate was roasted and the calcine cyanided. This resulted in a marginal (about 2%) increase in recovery at Lake Shore.

At Lake Shore the ore carries gold and silver. The gold is characterized by its extremely small particle size and by its presence to the extent of about 20% in combination with tellurium. Fine grinding of the ore is essential. Sulphides, chiefly pyrite, constituted 1½ to 2% of the ore.

Mining Methods

As mining progresses to deep levels, Lac is working on a variety of measures to provide more stable conditions, including new support systems, backfill techniques and seismology testing of high stress areas.

Annual Tonnage Milled/Ore Grade

For the full year of 1984 Macassa mined and milled 122,000 tons of ore with an average grade of 0.478 ounces of gold per ton. An additional 46,000 tons of ore at unstated grade, from the Lake Shore mine, was treated in the Macassa mill.

Total production from Lake Shore, including ore treated in the Terrains Auriferes mill, amounted to 65,000 tons with a recovered grade of 0.58 ounces per ton.

Gold Recovery, %

Average gold recovery rate for the Macassa ore in 1984 was 96.7%.

Gold Production

In 1984 gold production from Macassa was 58,472 ounces, not counting the Lake Shore contribution.

Total gold production from the Lake Shore crown pillar and other sources was 37,944 ounces.

Milling Process

The Macassa mill employs conventional cyanidation of the total ore. The process consists of the following:

Two-stage crushing to 3/8 in. or possibly coarser.
Grinding in two-stages to 95% minus 325 mesh.
Preaeration of pulp.
Thickening.
Cyanidation - single stage.
Filtration - single stage.
Gold recovery by Merrill-Crowe process.
Tailings pumped to tailings pond.

Operating Data

Refer to notes of visit in the Appendix.

Refractory Ores

The ore is not refractory but a small percentage of the gold may be present in solid solution with pyrite, unrecoverable except by roasting.

Comments

Macassa probably grinds its ore finer than any gold mill in Canada. Lime consumption is very high, apparently due to the high surface area of pyrite that results from the fine grinding.

Outlook

Macassa ore is high grade and with a new shaft will likely operate for several more years.

References

1. The Milling of Canadian Ores, CIM Congress Vol., 1957, Pp. 142,143.
2. Milling Practice in Canada, CIM, Vol. 16, Pp. 78, 79.
3. Northern Miner, Jan. 17, 1985, Pp. A1, A2.
4. Lac Minerals Ltd., Annual Report, 1984.
5. Notes of meeting with George Hope (see Appendix).

McBEAN MINE

Location

The McBean mine is located near the village of Dobie, 10 miles east of Kirkland Lake, Ontario.

Ownership

McBean is a joint venture between Inco Ltd. (65%) and Queenston Gold Mines Ltd. (35%).

Operating Costs

Not available.

History

The McBean open pit gold mine went into operation in March, 1984. Ore is processed in the old Upper Canada Mill, which was rehabilitated after being idle for 11 years.

Geology and Mineralogy

Gold occurs in zones related to the Larder Lake break. The ore is free milling.

Mining Methods

The ore is mined by open-pit methods.

Annual Tonnage Milled/Ore Grade

From published figures available it is estimated that 125,000 tons of ore were milled with an average grade of 0.10 to 0.12 ounces per ton.

Gold Recovery, %

In the range of 91-95%.

Gold Production

Estimate 13,000 ounces from March to December, 1984.

Milling Process

Conventional process with straight cyanidation of the ore and gold recovery by zinc precipitation. Free cyanide in barren bleed solution is destroyed by the Inco sulphur dioxide - copper sulphate process. This circuit operates about 1/3 of the time and is said to be quite effective. Control of pH in the process is critical.

Refractory Ores

The ore is not refractory.

Outlook

Open pit ore reserves will be depleted within the next year. Some additional ore may be recovered by going underground if the price of gold is high enough to warrant it.

References

1. Envers, Peter. "Gold bar pour heralds Kirkland Lake Revival," CMJ, July, 1984, Pp. 23-26.
2. Northern Miner, November 15, 1984. P. 12.
3. Visit to mine, March 28, 1985.

OWL CREEK MINE

Location

The mine is located 2 miles west of Kidd Creek Mines plant near Timmins, Northern Ontario.

Ownership

The mine is owned and operated by Kidd Creek Mines Ltd.

Economic Metals

Gold

Ore Reserves

Proven and probable reserves at year-end 1984 were 800,000 tons with an average grade of 0.12 ounces per ton.

Operating Costs

Not available.

History

Production at the mine began early in 1982 and has continued since that time.

Geology and Mineralogy

Host rocks are carbonaceous sediments. The gold is finely divided and is mostly associated with pyrite. Some carbon occurs in the ore.

Mining Methods

The ore is mined by open pit methods.

Annual Tonnage Milled/Ore Grade

The ore is shipped to Pamour's Schumacher mill for processing. In 1984, 294,000 tons of ore with a recovered grade of 0.084 ounces per ton were milled.

Gold Recovery, %

Gold recovery in 1984 was "somewhere in the 80s."

Gold Production

In 1984 gold production was 24,850 ounces.

Milling Process

The ore is treated in Pamour's Schumacher mill, described elsewhere in this report.

Types of Refractory Ore

The ore is somewhat refractory due to the fine dissemination of gold and the presence of carbon (graphite) in the ore, some of which is an active absorbent for gold in cyanidation.

Percent of Gold in Refractory Ore

Effectively, all the ore is somewhat refractory.

Current Status of Treatment of Refractory Ore

In the Schumacher mill the carbon is floated with the copper concentrate, which is shipped to a smelter. The pyrite is finely ground to recover as

much gold in cyanidation as practically possible.

Problems Related to Presence of Refractory Ores

As long as the active carbon is removed in the copper flotation circuit, there should be no particular problem in processing the Owl Creek ore.

Outlook

The mine will probably have a fairly short life unless reserves are expanded laterally, because the ore is probably too low in grade to justify underground operations.

References

1. Canadian Mines Handbook, 1984-85 P. 208.
2. Kidd Creek Mines Ltd. Annual Report, 1984.
3. Personal communication with C. Chapman.

PAMOUR PORCUPINE MINES LIMITED

Location

Pamour operates two milling plants. The No. 1 mill is located at the Pamour No. 1 mine about 20 km. east of Timmins, Ontario. The Schumacher mill, formerly owned by McIntyre Porcupine, is located about 3 km. east of Timmins.

The mills process ore from mines owned or leased by Pamour and, on a toll basis, from other mines in the area.

Ownership

Pamour is controlled by Noranda Mines Ltd. which holds 48.7% of the company's shares.

Economic Metals

Gold and silver. Some reserves have copper.

Operating Costs

In 1984, 84% of the ore treated in Pamour mills came from its own mines. Operating costs, including depreciation and development costs, amounted to $40.31 per ton of ore or $586 per ounce of gold produced. (Revenues from silver and copper were very small compared to gold.)

History

Production at the No. 1 mine began in May, 1936 at a rate of 750 tons per day. Since that time the rate was progressively increased to 3,000 tons/day.

In 1973 Pamour purchased McIntyre Mines property and plant which have since been operated as Pamour's Schumacher division. In 1976 Hollinger's property at Timmins and its Ross Mine were acquired.

Since the late 1960's a number of other properties in the Porcupine and Matachewan areas were purchased or leased by Pamour, including Buffalo Ankerite, Falconbridge-Hoyle, Peninsular, Aunor, Hallnor, Coniaurum and Delnite. Ores from these mines have been treated in the two central mills of Pamour.

In 1984 Pamour operated the Pamour No. 1 mine, the No. 3 (Aunor) mine and the Schumacher, Ross and Timmins mines. It is believed that ore from the first two mines went to the No. 1 Mill and from the others to the Schumacher Mill. In October, 1984 operations were suspended at the Timmins Underground mine and at the No. 3 mine due to declining gold price and failure of the Timmins mine to meet expectations.

Geology and Mineralogy

The gold ore of the Porcupine area can be broadly described as quartz and quartz-ankerite veins filling fractures in highly altered, and predominantly intermediate lavas, quartz porphyries and sediments.

Gold is present in sulphides and in quartz. When in quartz it is distributed irregularly along fractures as flakes and scales. Much of the gold is intimately associated with pyrite as films or flakes in fractures and cleavage planes of the crystals or as inclusions within the crystal.

The ratio of gold to silver varies from mine to mine. Most of the silver is present as an alloy with gold. Tellurides of gold and silver have been identified but are extremely rare.

The most abundant sulphide is pyrite, representing from 2 to 8% of the ore.

At the Pamour No. 1 mine gold occurs in disseminated zones and in veins in volcanics and sediments. The disseminated zones are stockworks of quartz stringers, accompanied by pyrite, pyrrhotite and small amounts of gold. Veins in volcanics generally carry free gold and very minor amounts of sulphides. Veins in sediments are accompanied by "bleaching" and sulphides.

At the Ross Mine stockworks of narrow auriferous quartz veins occur in breccias and lavas in which sericite, carbonate, quartz and pyrite were developed before deposition of the veins. The ore contains a small percentage of chalcopyrite.

The gold zones at Schumacher are in a series of steeply dipping lava flows which have been intruded by quartz-feldspar porphyry. The gold occurs in numerous quartz-carbonate lenses in an area about 7,000 feet in length by 4,000 feet in width to a depth of 7,500 feet. The amount of quartz present varies considerably and ranges from 100% of the vein material to a network of widely separated stringers. Sulphide mineralization, predominantly pyrite, is present in varying degrees both in the quartz and in the wall rock adjacent to it. Gold occurs associated with the sulphides and scattered independently throughout the veining, in which case it may be readily visible.

Some pyrrhotite occurs in the Schumacher ore and the content has been increasing with depth. The presence of appreciable amounts of graphite in the ore, some of it an active absorbent for gold, plagued McIntyre metallurgists for years.

A completely separate zone at Schumacher contains gold and chalcopyrite. It was mined from 1967 to 1982.

Mining Methods

At Pamour No. 1 shrinkage stoping, cut-and-fill open stoping and sub-level shrinkage stoping have been used. Vertical crater retreat stoping has been introduced at this mine and the Ross Mine as well.

Some open pits are probably still being mined.

Annual Tonnage Milled/Ore Grade

Tonnages and grades of ores treated in the two mills in 1984 were as follows:

(Totals include very minor production from other sources, not listed in table)

	No. 1	No. 3	Schum.	Ross	Timmins	Total
Ores milled - tons (000's)	647	59	275	237	289	1,509
Average rate per day - tons	1,774	161	754	649	792	4,123
Ore Grade						
Gold - oz./ton	0.069	0.118	0.082	0.094	0.080	0.079
Copper - %	-	-	-	0.120	-	0.020

Gold Recovery

Gold recoveries averaged 88.5% and 85.3% respectively for the Pamour and Schumacher mills.

Gold, Silver and Copper Production

Gold and silver production in the form of bullion and concentrate, and copper recovered in concentrate in 1984, were as follows:

	No. 1	No. 3	Schum.	Ross	Timmins	Total
Gold -oz.	40,412	6,197	20,042	17,875	19,174	103,804
Silver-oz.	11,910	-	10,122	39,926	-	61,958
Copper-tons				173		173

Milling Processes

The two Pamour mills are generally similar, but the Pamour No. 1 mill treats free milling ores whereas the Schumacher treats copper-bearing and/or carbonaceous ores.

The process in the No. 1 mill consists essentially of the following:

Primary-jaw crushing underground

Two-stage cone-crushing in the mill

Two-stage ball milling

Primary bulk flotation

Regrinding and flotation of tailings
Tailings pumped to tailings pond
Primary and secondary concentrates cleaned once
Cleaner concentrates reground and thickened
Two-stage preaeration in lime slurry
Four-stage cyanidation and five-stage counter-current wash thickening
Gold recovered from pregnant liquor by Merrill-Crowe process
Leached solids and barren solution pumped to a separate lagoon with no overflow

The Schumacher milling circuit includes the following operations:
Primary crushing at the mines
Two-stage cone crushing in the mill to 7/16 in.
Two-stage ball milling
Copper-gold flotation with two or three cleaning stages
Copper-gold concentrate dewatered and shipped to smelter
Pyrite flotation from copper tailings
Pyrite flotation tailings classified to produce mine backfill and slime product, which is pumped to tailings pond
Pyrite-gold concentrate regrinding
Prefloat of reground concentrate with concentrate recycled to copper circuit
Pyrite-gold flotation
Aeration of pyrite-gold concentrate in lime
Two-stage cyanidation and two-stage filtration of leached solids
Recovery of gold in pregnant solution by Merrill-Crowe process
Acid treatment of precipitate to remove copper and zinc (copper recovered) prior to smelting to produce bullion.

Types of Refractory Ore

Ores treated by Pamour are refractory because of the fine dissemination of gold in sulphides and/or the presence of active graphitic (carbonaceous)

material in the mill feed. The graphitic ores are treated in the Schumacher mill.

Percent of Refractory Ore to Total Ore

Pamour treats ore from various sources, and feed to the two mills varies from time to time. However, it appears that most of the ore treated is somewhat refractory to treatment.

Problems Related to Presence of Refractory Ores

Pamour has problems in blending ores at the Schumacher mill in order to maintain an acceptable copper grade in the copper-gold concentrate without allowing active graphite into the cyanidation circuit. Some difficulty has been reported with non-commercial galena and sphalerite reporting in the copper concentrate.

At the Pamour No. 1 mine, potassium permanganate has been used in concentrate grinding and pre-aeration to promote oxidation of pyrrhotite.

Comments

Tests have shown that recovery could be improved at Pamour mills by cyaniding the flotation tailings and recovering the gold by carbon adsorption. However, Pamour has concluded that, because of the low grade of ore treated (less than 0.10 ounces per ton) cyanidation of the flotation tailings is not economically feasible at present.

Outlook

All Pamour's mines are marginal operations and some have been closed because of operating losses. By increasing production from the mines still operating and by treating custom ores, Pamour hopes to keep its two mills operating at capacity in the short term at least.

References

1. The Pamour No. 1 Mine, CIM Reporter, Sept. 21, 1980, P. 8.
2. Milling Practice in Canada, 1978, Pp. 69-76.
3. Pabst, Marilyn. "Activity at Pamour - exploration up, reserves up, possible reopenings." CMJ. Dec. 1979, Pp. 30-33.
4. CMJ, Sept. 1976, Pp. 54-58.
5. Raicevic, D. and Bruce, R.W. "Gold recovery from a refractory carbonaceous gold ore" CMJ, March, 1976, Pp. 40-45.
6. Pamour Porcupine Mines Ltd. Annual Report, 1984.
7. Wagner, R.H.O. "The economics of the carbon in pulp treatment of low grade flotation tailings." CMP. Proceedings, 1984, Pp. 344-368.

RENABIE MINE

Location

The Renabie Mine is located at Missanabie, near Wawa in Northern Ontario.

Ownership

Royex Gold Mining Corporation, through Cullaton Lake Mines has a 50% interest in Renabie Gold Mines Ltd. The other 50% is owned by Barrick Resources Corporation. Royex is part of the Campbell Resources Inc. group and manages the Renabie Mine.

Economic Metals

Gold

Ore Reserves

As at June 30, 1984 estimated proven, probable and drill indicated reserves were approximately 1.5 million tons averaging 0.19 ounces of gold per ton, according to Royex.

Operating Costs

Barrick and Campbell didn't report individual mine operating costs in 1984.

History

The Renabie mine commenced operations in 1947 and operated until 1970. During this period 3,686,000 tons of ore were milled with an average grade of 0.21 ounces of gold per ton, yielding 780,000 ounces of gold. Mill head grade was reported to average 0.23 ounces per ton and mill recovery averaged

94% during the last seven operating years.

Sungate Resources took over the mine in 1979 and resumed production in January,1982. The present owners acquired the mine in October, 1983.

The production rate dropped in 1984, reportedly due to a change-over to a more selective mining method and to make repairs to the headframe.

Geology and Mineralogy

The ore is mainly white quartz carrying very fine particles of gold. The sulphide content is fairly low, probably less than one percent. Pyrite is the most abundant sulphide but some galena occurs in some orebodies. Minor amounts of pyrrhotite, chalcopyrite, chalcocite and sphalerite have been noted. Gold occurs in finely divided form in the pyrite as well as in the quartz.

Mining Methods

Sungate used long hole open stoping. The present owners are changing over to cut-and-fill which is a more selective method of mining.

Annual Tonnage Milled/Ore Grade

For the first nine months of 1984, 98,605 tons were milled with an average grade of 0.15 ounces per ton.

Gold Recovery, %

Average mill recovery for the same period was 92.7%

Gold Production

In the nine month period 13,325 ounces of gold were produced.

Milling Process

When the plant first started the ore was ground to 75%-200 mesh and floated. The flotation tailings were discarded and the concentrate ground to at least 98% minus 325 mesh and cyanided for a 96 hour period. At that time aeration of the concentrate with lime, prior to cyanidation, was practiced but was abandoned when it was found to be detrimental.

Jigs for the removal of free gold were introduced into the grinding circuit in 1948 but were replaced by unit flotation cells in 1952.

In 1956 the plant was altered to permit cyanidation of the total ore, followed by flotation of the tailings to recover pyrite, which was recycled to the grinding circuit.

The milling circuit, as reported in 1961, consisted of the following units:

Conventional crushing,
Open circuit rod milling,
Pebble milling in closed circuit with mechanical classifiers (2 parallel circuits, in cyanide solution),
Thickening (4 units in parallel),
Cyanide leaching in two stages,
Filtration in two stages,
Flotation of leached solids to recover pyrite,
Pyrite reground in a pebble mill and recycled to cyanidation,
Waste solids pumped to tailings pond,
Gold recovery from solution by Merrill-Crowe process.
It is believed that the basic flowsheet hasn't changed since 1961.
Nominal milling capacity was 500 TPD.

Refractory Ore

The ore is not refractory but the total ore must be finely ground, and the sulphides extra finely ground to attain high recovery of gold.

Outlook

An expansion program is underway to increase annual gold production to approximately 60,000 ounces per year. The plan contemplates the sinking of

an internal shaft to provide access to mine levels between 3,100 and 4,500 ft. levels, expansion of the mill to 850 tons per day and changing to a cut-and-fill mining method.

References

1. Kearney, E.G. "The First Two Years of Milling at Renabie," CIM Bull. Jan. 1950, Pp. 37-44.
2. The Milling of Canadian Ores, CIM Congress Vol. 1957, Pp. 153-154.
3. Joint Management Information Circular, Shareholders Meetings of Royex Gold Mining Corporation and Pezamerica Resources Corporation, Jan. 18, 1985.
4. Campbell Resources Inc. Annual Report, June 30, 1984.
5. Royex Gold Mining Corporation, Interim Report, Dec. 31, 1984.
6. Canadian Mining Manual, 1961, P. 86.

SCADDING GOLD MINE

Westfield Minerals Limited in 1984 completed the first phase of a mining operation on the Scadding property near Sudbury, Ontario. From two surface deposits, 24,000 tons of ore averaging 0.2 ounces per ton were treated in a 200 TPD mill. Gold production in flotation and gravity concentrates amounted to 3,500 ounces. Predicted recovery was 90%.

The feasibility of underground mining is under review and largely depends on the price of gold. Ore reserves before production commenced in June were 205,000 tons at an average grade of 0.234 ounces per ton.

References

1. Western Miner, May, 1984, P. 15.
2. Northern Miner, June 21, 1984.
3. Northgate Exploration Ltd. Annual Report, 1984.

CAROLIN MINE

Location

Carolin's Ladner Creek property is located near the town of Hope, about 100 miles east of Vancouver, B.C.

Ownership

The Ladner Creek mine is a 50/50 joint venture between Carolin Mines Ltd. and the Aquarius Group.

Economic Metals

Gold

Ore Reserves

Ore reserves were reported to be 800,000 tons with an average grade of 0.13 ounces per ton, late in 1984.

Operating Costs

Recent figures are not available but Carolin has been losing money for some time, with operating costs obviously well above the revenues received from gold.

History

Carolin's 1500 TPD mill commenced operations in late 1982. From the beginning, the mine experienced problems particularly with its cyanide destruction process. The mine was shut down on at least one occasion as a result of cyanide spills into tributaries of the Fraser River. Gold recoveries in the mill failed to meet projections. Operations were suspended, on or about September, 1984.

Geology and Mineralogy

Most of the gold occurrences are hosted by sedimentary rocks along the edge of a serpentine belt.

The ore consists of roughly equal amounts of quartz, calcite and albite. Average sulphide content reported ranges from 6 to 15% by volume. The main sulphides are pyrrhotite, pyrite and arsenopyrite. Gold is associated with these sulphides, usually along the rims of grain boundaries and usually accompanied by traces of chalcopyrite. Occasionally visible free gold is seen, mainly smeared along slickensides.

Mining Methods

The underground mine is developed from two adits and a decline.

Annual Tonnage Milled/Ore Grade

No information available for 1984.

Gold Recovery, %

Mill recoveries have been poor, apparently less than 80%.

Gold Production

Figures for 1984 are not available, however it was reported in September, 1984 that 39,908 ounces of gold had been produced since the mine opened in 1982.

Milling Process

Carolin employed a milling process that was probably unique in Canada. Following conventional crushing and grinding, a concentrate containing sulphides and carbonaceous material (graphite?) was recovered by flotation. Primary flotation tailings were pumped to the tailings pond.

The primary concentrate was leached in cyanide and a secondary flotation concentrate recovered by flotation of the leach slurry. This secondary concentrate, containing carbonaceous material, was batch-treated to recover gold held by adsorption on the solids. The secondary flotation tailings were thickened and filtered twice. Solids were repulped and pumped to tailings. Dissolved gold in the thickener overflow and filtrates was recovered by the Merrill-Crowe process. Barren solution was discarded following treatment by alkaline chlorination or the Inco process to destroy cyanide compounds.

Effluent from the tailings pond was also treated, but by alkaline chlorination, to destroy cyanide before being discarded into Ladner Creek.

Milling Problems

Carolin has experienced a number of milling and environmental problems. Detailed descriptions of the problems don't appear to have been published.

The original engineering report on Carolin specified only 80% gold recovery in milling. Gold losses may have been attributed to gold adsorbed on carbonaceous material not recovered in secondary flotation or possibly to finely disseminated gold or gold in solid solution in one or more sulphides. Mill recoveries apparently fell short of predictions.

Pyrrhotite in the ore resulted in high cyanide consumption and in the formation of iron cyanide compounds during leaching. For some time after the mill started, the barren solution was treated by alkaline chlorination to destroy cyanide compounds. This system was apparently less effective than expected and, of course, didn't destroy the iron cyanide compounds. As a result, there was a built-up of cyanide complexes in the tailings system.

The mine was closed for a total of four months in 1982 and 1983 due to a cyanide spill into two tributaries of the Fraser River.

In October, 1983 barren treatment using Inco's process replaced alkaline chlorination. The effluent from the tailings pond was treated separately by alkaline chlorination.

Early in 1984 Carolin abandoned cyanidation entirely and produced a sulphide concentrate for shipment to a smelter.

Due to environmental problems and heavy financial losses, Carolin suspended production on or about September, 1984.

Outlook

Carolin is unlikely to resume operations unless there is a substantial increase in the price of gold or higher grade ore is discovered.

References

1. Cochrane, D.H. "Developing Carolin Mines - an uphill battle," CMJ April, 1980, Pp. 35-38.
2. Barr, D.A. "Gold in the Canadian Cordillera," CIM Bull, June, 1980 Pp. 59-76.
3. Northern Miner, Sept. 13, 1984, Pp. 1,2; Nov. 22, 1984, P. 21; Jan. 3, 1985, Pp. 1,2; Feb. 14, 1985, P. 18.
4. Hoi-Yen, Der, Financial Times, Sept. 24, 1984.
5. Shearer, J.T. and Niels, R.J.E., "Carolin Mines - a geological update. West. Min. Nov. 1983, Pp. 21-24.
6. Personal communication, B. Conard.

EQUITY SILVER MINES

Location

Sam Goosly property, mine and mill, is located 37 Km south of Houston in north-central B. C.

Ownership

Placer Development Limited has 70% interest in Equity.

Economic Minerals

Silver, plus some gold and copper.

Equity is Canada's largest silver mine.

Ore Reserves (Mineable), December 31, 1984

	Tonnes	g/t Ag	g/t Au	% Cu	Strip Ratio
Main Zone	16,756,000	110.0	0.99	0.37	3.8
Waterline Zone	2,526,000	93.4	1.39	0.38	3.6
Total	19,282,000	107.8	1.04	0.37	3.8

Reserves based on cut-off grade of 90 g/t Ag equivalent.

A small high grade zone beneath the Southern Tail pit is not included in reserves.

Operating Costs

"Cost of sales" in 1984 was $51,676,000, equivalent to $24.72 per tonne of ore milled.

History

Milling commenced in October 1980 on ore from the Southern Tail Zone. Start up problems in the complex leaching circuit, designed to remove arsenic and

antimony from flotation concentrate afflicted the operation and acceptable levels of throughput weren't achieved until March, 1983.

In 1983 mining commenced in the Main Zone orebody and in March 1984 the company completed the switch of mining operations from the Southern Tail Zone to the Main Zone. The lower grade and more finely mineralized ore of the Main Zone required the addition of grinding, flotation, dewatering and other equipment to maintain capacity. This was added during 1983 and became operational in July of that year.

Leaching operations were suspended in April, 1984. This became possible because of lower impurity levels in the Main Zone ore, improved smelter terms and greater market acceptance of the unleached concentrate.

A scavenger circuit to recover gold and silver from current concentrator tailings commenced operations in January, 1985.

Approximately 7 million tonnes of tailings from the Southern Tail Zone have been accumulated. This material averages 20 g/t silver in unliberated tetrahedrite and/or minor silver-bearing sulphosalts, and 0.9 g/t gold associated with arsenopyrite. A 2 t/d pilot plant to recover precious metals from these tailings by bioleaching went into operation recently.

Geology and Mineralogy

The mine is underlain by diverse assemblages of Jurassic to Eocene volcanic and sedimentary rocks which are intruded by Tertiary intrusives. Stockwork zones contain disseminated and massive sulphides that are rich in pyrite, chalcopyrite and tetrahedrite. Other sulphide minerals include pyrrhotite, sphalerite, galena, and arsenopyrite.

Silver occurs in tetrahedrite and, to a lesser extent, in other minerals.

There were two major deposits: the Southern Tail and the Main Zone. Mining started on the Southern Tail because it contained higher metal concentrations and had softer ore. The Main Zone contains less antimony and arsenic but the sulphide minerals are more finely disseminated than in the other zone, and the ore is less amenable to concentration by flotation.

Mining Methods

Open pit mining. Development and mining of the Main Zone was at an initial strip ratio of 5.3 tonnes of waste and low grade ore to 1 tonne of ore.

Annual Tonnage Milled/Ore Grade

Milled 2,090,000 tonnes in 1984, at an average daily rate of 5,700 tonnes. Average grade of mill feed was 113 grams per tonne of silver. Gold and copper contents were not reported.

Metal Recoveries

Calculated silver recovery in concentrate: 61%
Gold recovery not reported. Estimate: 42%
Copper recovery not reported but appears to be high.

Metal Production (In Concentrate), 1984

Silver	143,820 Kg (4,624,000 oz.)
Gold	740 Kg (23,800 oz.)
Copper	9,020 t. (19,882,000 lb.)

A total of 7,900 tonnes of leached concentrate was produced before the leach plant was shut down.

Milling Process

The leaching process, now abandoned, utilized sodium sulphide and caustic soda to extract antimony and arsenic from flotation concentrate. Following treatment the concentrate was sold to a smelter. By-products from the leaching process included sodium antimonate, sodium sulphate and calcium arsenate. The first two products were marketed but the arsenate was trucked away from the site for disposal. This complex leaching process is well described in the literature.

The present process consists of conventional crushing and grinding circuits, a flotation circuit to recover a silver-gold-copper concentrate, and a scavenger

cyanidation circuit to recover additional gold and silver from the flotation tailings. This scavenger circuit, which employs carbon-in-pulp leaching followed by Merrill-Crow precipitation of precious metals, is described in the Canadian Mining Journal, November, 1984, Pp. 22-25.

Tests in 1983 indicated that the scavenger circuit would recover over half the 0.45 grams per tonne of gold and one-third of the 30.0 grams per tonne of silver in the flotation tailing. (Assuming mill feed is equal to average grade of Main Zone reserves, these figures indicate improved recoveries of gold and silver of over 25% and of 9% respectively.)

The bioleaching pilot plant includes bulk flotation of the tailings, pre-oxidation of the concentrate using bacteria, liquid/solids separation followed by carbon in leach to recover the precious metals.

Types of Refractory Ore

Silver losses are believed to be mainly associated with unliberated tetrahedrite whereas gold losses are mainly associated with arsenopyrite. (Arsenopyrite recovery in flotation is undesirable.)

Percent of Refractory Ore to Total Ore

Effectively 100%.

Percent of Precious Metals Present in Refractory Ore

Effectively 100%.

Current Status of Treatment of Refractory Ores

As described above.

Problems Related to Treatment of Refractory Ores

The presence of antimony and some arsenic in the mill feed presents a challenge to metallurgists to produce a flotation concentrate that is acceptable to smelters, while at the same time maximizing metal recoveries.

Outlook

Sales of concentrate will depend on continued acceptance of the concentrate by smelters.

The outlook for the operation will be much clearer after the mill has operated for a full year on Main Zone ore with the scavenger cyanidation circuit in operation.

References

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2. Sorensen, Jean. Carbon-in-leach circuit to up silver-gold recovery at Equity. CMJ Nov. 1984, Pp. 22-25.
3. Wide-ranging interests of Placer Development, Western Miner, April, 1984, P. 9.
4. Lithogeochemistry at the Equity Silver Mine, Western Miner, April, 1984, Pp. 50-54.
5. CMJ March, 1985, P. 17.
6. Equity Silver features unique leach process. CMJ, Aug. 1981, Pp. 54-58.
7. Edwards, C. R., and Melville, R. G. The Equity Leach Plant Startup. Proc. of the 14th Annual Meeting of the Cdn. Min. Proc., 1984, Pp. 251-272.
8. Cobban, G. Enthusiasm at Equity Silver as start up nears. Western Miner, July 1980, Pp. 19-22.
9. Equity Silver: complex metallurgy resolved. Western Miner, Aug. 1981, Pp. 11-18.

ERICKSON GOLD MINE

Location

The Erickson operations are located on Table Mountain, 7 miles northeast of Cassiar in Northern B. C. Erickson's claims cover ninety square miles in the McDame Creek area.

Ownership

Erickson Gold Mines Ltd. is an independent company.

Economic Metals

Primarily gold.

Ore Reserves

In January, 1984 proven and probable ore reserves were 147,000 tons averaging 0.48 ounces per ton. In late 1984 the company reported that ore reserves were being maintained.

Operating Costs

Details not available but Erickson reported an operating loss of $1.4 million for the first nine months of 1984.

History

The first sustained milling operation at Erickson commenced in December 1978 and full production in the 100 TPD mill was reached in April 1979. In January 1983 the mill was expanded to 200 TPD and later to 265 TPD.

Initial production came from the Jennie vein and several other veins in the vicinity. During 1983 Erickson acquired the assets of Table Mountain Mines Ltd. and most of the assets of Plaza Mining Co. These properties contain the Vollaug Vein, which has a strike length of at least 8,500 ft.

Geology and Mineralogy

The original Erickson deposit is a mesothermal Bralorne type. Multiple vein systems occur adjacent to major through-going linear fault structures. Multi-stage mineral injection has resulted in a concentration of gold and silver values.

Indications are that the Vollaug vein and Plaza properties are similar.

Much of the gold occurs as coarse particles.

Mining Methods

Open pit and underground mining methods are employed.

The underground mine is developed from a series of adits, each new one progressively lower down the mountain. A modified form of shrinkage stoping is used.

Annual Tonnage Milled/Ore Grade

For the first nine months of 1984 approximately 70,000 tons of ore was milled, with an average grade of 0.33 ounces per ton.

Gold Recovery, %

Historical recovery is 95%.

Gold Production

For the first nine months of 1984 gold production was 21,000 ounces.

Milling Process

The milling process is quite simple. The ores are crushed and ground by conventional methods. The ore is free milling and over 60% of the gold is recovered by jigs in the grinding circuit. The remaining gold is recovered by flotation and the concentrate is shipped to a smelter in Montana.

There are wide variations in the grade of ore treated but this has no appreciable affect on the gold recovery.

Refractory Ore

No refractory ore types occur in the deposits.

Outlook

Erickson has large land-holdings in the area but the outlook is uncertain. In the first three quarters of 1984 the company reported a loss of $1.4 million but operations were profitable in the third quarter.

References

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3. Northern Miner, May 17, 1984 and November 15, 1984, P. 23.
4. Western Miner, February, 1984, P. 67.
5. Canadian Mines Handbook, 1984-85, P. 139.

SCOTTIE GOLD MINE

Location

The mine is located at Summit Lake, 30 miles north of Stewart, in Northwestern British Columbia.

Ownership

Northair Mines Ltd. is the major shareholder (19%) and manages Scottie.

Economic Metals

Gold, minor silver.

Ore Reserves

Proven reserves were reported to be 117,000 tons at average grade of 0.55 ounces per ton, in September, 1984.

Operating Costs

Costs were about US $370. per ounce of gold produced late in 1984.

History

Mine development began in 1979. In October, 1981 a 200 TPD mill went into operation.

Geology and Mineralogy

No information.

Mining Methods

Probably use shrinkage stoping.

Annual Tonnage Milled/Ore Grade

In the year ending September 30, 1983, Scottie milled 67,415 tons at a recovered grade of 0.51 ounces per ton.

Gold Recovery, %

Typical recovery at Scottie was 93%.

Gold Production

For the year ending September 30, 1983 gold production was 34,090 ounces.

Milling Process

The process consisted of crushing, grinding, pre-aeration, cyanidation of the total ore and gold recovery by the Merrill-Crowe process.

Refractory Ore

The ore is not refractory.

Outlook

Operations were suspended about the end of January, 1985 because the price of gold was too low to continue operations. The mine and plant are being maintained.

References

1. CMJ December, 1979, P. 45.
2. Canadian Mines Handbook, 1984-85, P. 339.
3. Northern Miner, Jan. 24, 1985, P. 13.
4. Financial Times, Feb. 4, 1985, P. 3.
5. Personal communication, J. Barrett.

COMINCO'S CON OPERATIONS

Location

Cominco's Con operations, including the Con and Rycon mines, are located on the south-east outskirts of Yellowknife, N.W.T.

Ownership

Con is an operating property of Cominco Limited.

Economic Metals

Primarily gold. High grade arsenic trioxide is also produced.

Ore Reserves

Measured and indicated reserves in 1984 were 1,700,000 tons with an average grade of 0.42 ounces of gold per ton.

Operating Costs

Con's revenue in 1984 was $44 million and operating profit $5 million. Therefore, calculated operating cost was $39 million, equivalent to approximately C$ 440 per ounce of gold produced. (Arsenic plant revenue and profit are included in the above figures. If excluded, the operating cost per ounce of gold would probably be somewhat lower.)

History

The Con mill commenced operations in 1938 at a rate of 100 TPD using amalgamation and cyanidation. It has operated almost continuously since except for three years during World War II.

The mill has been expanded several times to its current rate of 750 TPD and process changes have been made to keep pace with mineralogical changes in the ore. Increasing proportions of gold-bearing arsenopyrite in the feed

led to the introduction of a flotation circuit in 1941 and an Edwards roaster and calcine leaching facilities the following year. Amalgamation was discontinued in 1968 when the gold became finely disseminated. A trend towards less arsenopyrite at lower mine levels resulted in a shut-down of the roaster and related facilities in 1970.

Currently, an increasing amount of coarse gold in the ore will probably lead to the re-introduction of gravity concentration. Flotation of free gold is also being studied.

A plant to recover arsenic from an old roaster dust pond was commissioned in February 1984, following major revisions and was operating reliably at year-end.

Geology and Mineralogy

Gold deposits at Yellowknife occur in greenstones comprising volcanic and sedimentary rocks. The principal rock is quartz with the balance being carbonate, carbonate-sericite schist or carbonate-chlorite schist.

Currently most of the gold occurs in the native form and is quite coarse - about 75% coarser than 150 mesh (150 microns). The gold occurs in quartz and also in association with pyrite, arsenopyrite and sulphosalts. The proportion of gold in intimate association, or in solid solution with arsenopyrite has never approached that at the nearby Giant mine. At present no attempt is made to recover the gold locked in arsenopyrite.

Mining Methods

Work is progressing on deepening the main Robertson shaft by 810 feet to the 6,235-foot level, to be completed in 1986. Most of the ore is believed to be mined by cut-and-fill methods.

Annual Tonnage Milled/Ore Grade

In 1984 ore milled amounted to 244,000 tons with an average grade of 0.39 ounces per ton.

Gold Recovery %

Operators reported in September, 1984 that leach extraction was 90-92% and there was a soluble loss of 0.5%.

Cominco did not report the % gold recovery in its 1984 report but the production figures indicate recovery of 93.5% plus or minus 1%. This includes an unstated amount of gold recovered in the arsenic plant.

Gold Production

Gold production for 1984 was 89,100 ounces including gold recovered in the arsenic plant.

Additionally 1,400 tons of high grade arsenic trioxide was produced.

Milling Process

The mill was recently rehabilitated but the straight cyanidation flow-sheet unchanged. It consists of the following:

Primary crushing underground.

Jaw and cone-crushing to 1/2 in.

Two-stage ball milling, both in closed circuit with cyclones, to 75%-200 mesh, in cyanide solution.

Thickening to 60% solids.

Three-stage cyanide leaching, with drum filtration following each stage. Retention time 48 hr.

Gold recovery by Merrill-Crowe process.

Waste solids classified once to produce mine backfill.

Fine solids pumped to tailings pond.

Water effluent treatment with ferric sulphate to precipitate ferrocyanide as Prussian Blue and precipitate arsenic.

A new arsenic plant to treat crude arsenic from an old waste pond uses a two-stage hot water leach to dissolve the arsenic. Following a liquid/solids separation the solution is cooled and arsenic trioxide recovered in a three-stage crystallizer circuit. The crystals are filtered out, dried and packed for shipment to the USA. The product contains a minimum of 99.5% arsenic trioxide.

Types of Refractory Ore

Refractory ore contains a small percentage of arsenopyrite with some gold in sub-micron form.

Percent of Gold in Refractory Ore

The unrecoverable gold in arsenopyrite is probably less than 5% of the total gold in the ore.

Percent of Refractory Ore to Total Ore

Most of the ore mined is not refractory.

Problems Related to Treatment of Refractory Ore

The Con Mine is adjacent to Yellowknife and the property extends into Yellowknife Bay. Con has, in the past, created environmental problems due to arsenic, possibly in-plant as well as externally. The Edwards roasters and Doyle wet scrubbers used when roasting was practiced, and disposal of dust in an outdoor pond would not be acceptable practice today. By discontinuing roasting, recovering arsenic dust from the storage pond, and treating waste water from the mine and mill to remove arsenic, Con appears to have resolved earlier problems with arsenic.

Comments

Like many gold mines, Con was slow to modernize their plant when the price of gold went up in the 1970s. However, in the last couple of years Con has worked aggressively to increase gold recovery and modernize their mill. In 1984 Con was considering various methods of gravity concentration as well as flotation to increase gold recovery or reduce costs.

Con has installed some Atara "Cannon" mixers to replace some of its old air-lift agitators. A determined program was also underway in 1984 to adapt a Tower grinding mill to treat hard ores and produce a very fine product.

Outlook

Con is now one of the deepest mining operations in Canada. However, ore reserves are sufficient to keep the mine in operation for seven years at the current operating rate, if the price of gold at least keeps pace with operating costs.

References

1. Edwards, I. C. Report to CANMET on Visits to Gold Mines in the Northwest Territories, Sept. 1984.

2. Cominco Ltd. Annual Report, 1984.

CULLATION LAKE MINE

Location

The mine is located in the Northwest Territories, about 250 miles northwest of Churchill, Manitoba.

Ownership

Royex Gold Mining Corporation, a member of the Campbell Resources group, is the sole owner of the Cullaton Lake Mine.

Economic Metals

Gold

Ore Reserves

As of June 30, 1984 the original "B" zone deposit contained estimated proven and probable reserves aggregating approximately 89,600 tons averaging 0.49 ounces per ton. The new Shear Lake zone was estimated to contain probable and drill-indicated reserves of 955,000 tons with an average grade of 0.21 ounces of gold per ton. (These estimates were published by Royex. Campbell reported that Shear Lake reserves at the same time were 420,000 tons at an average grade of 0.22 ounces per ton).

Operating Costs

Not available but are apparently quite high.

History

The mine went into production in October, 1981 with production coming from the "B" zone underground mine. Serious technical problems were encountered with the breaking in of the mill, apparently due to problems in handling

frozen muck and unfamiliarity with the "state of the art" process. In 1982 Cullaton was reorganized and Royex assumed management control. On January 1, 1983 the mine was brought into "commercial production."

With dwindling reserves in the "B" zone, Cullaton in 1984 concentrated on developing the Shear Zone and made some modifications to the mill to treat ore from both mines.

Geology and Mineralogy

The "B" zone ore deposit consists of gold occurring in sulphides in an Archean iron formation. The banded iron formation is composed principally of four distinct mineralogical facies, namely carbonate, oxide, sulphide and silicate, hosted by a thick volcaniclastic sedimentary assemblage. The gold is confined to the sulphide facies and is directly associated with pyrite and pyrrhotite.

At the Shear Lake property, free gold occurs in highly oxidized pyritic quartzite along shear zones, in a completely different environment from the "B" zone.

Mining Methods

Both properties are developed from underground ramps.

Annual Tonnage Milled/Ore Grade

For the fiscal year ended September 30, 1984 Royex reported that 90,825 tons were milled with an average grade of 0.33 ounces per ton.

Gold Recovery, %

Average mill recovery for the same period was 92.9%.

Gold Production

Gold production for the fiscal year was 27,875 ounces.

Milling Process

The original milling process consisted of the following:

Primary jaw crushing to 6 inches,
Semi-autogenous grinding with 5 inch balls,
Ball milling in closed circuit with cyclones,
Thickening (high capacity type),
Preaeration and cyanidation in six mechanically agitated tanks with 52 hr. total retention time,
Gold recovery by carbon-in-pulp (5 stage),
Stripping, electrowinning and smelting of gold,
Wastes pumped to tailings pond.

Original test work indicated that preaeration would speed up the rate of gold dissolution in leaching and reduce cyanide consumption by 20 to 50 percent. The circuit was designed for 8.6 hours preaeration and 43.2 hours cyanidation but preaeration could be eliminated, or preaeration time doubled by switching the point of cyanide addition.

Leach testwork defined the required grind to be 80% passing 53 microns (270 mesh).

Some modifications were being made to the mill in 1984 to treat Shear Lake ore, and to expand the capacity to 400 tons/day.

The Shear Lake ore is unusual in that it consists of a highly brecciated quartzite wherein the matrix between particles contains the gold in association with pyrite, hematite and ilmenite. Tests have shown that by crushing the ore and separating the fines, the ore can be upgraded to approximately 0.45 ounces of gold per ton. This preconcentration process may be used on the Shear Lake ore.

Types of Refractory Ore

The "B" zone ore is not very refractory but requires very find grinding

to liberate the gold.

Detailed metallurgical data on the Shear Lake ore don't appear to have been made public.

Outlook

The decline on the Shear Lake property is being extended to the 150 m level. Cullaton hopes to prove up enough reserves in 1985 to justify a mill expansion of up to 1,000 tons per day. It is apparent that a higher rate of production is necessary, at the present price of gold, to make the operation viable.

References

1. Rawling, K.R., Mather, R.F. and Childs, A.M. "Process Design Considerations for a Small Gold Mill in the N.W.T. Proc. of Canadian Mineral Processors, 1983, Pp. 7-20.
2. Envers, P. "Cullaton Lake: Worst over as second mine starts up." CMJ June, 1984, Pp. 56-58.
3. Campbell Resources Inc. Annual Report, June 30, 1984.
4. Joint Management Information Circular, Shareholder Meetings of Royex Gold Mining Corporation and Pezamerica Resources Corporation, Jan. 18, 1985.
5. Royex Gold Mining Corporation, Interim Report, Dec. 31, 1984.

ECHO BAY'S LUPIN MINE

Location

The Lupin mine is located on the shore of Contwoyto Lake, some 250 miles NNE of Yellowknife, N.W.T.

Ownership

The mine is fully owned by Echo Bay Mines Limited of Edmonton.

Economic Metals

Gold, with small amount of silver.

Ore Reserves

Reserves at the end of 1983 stood at 3.4 million tons grading 0.396 oz. of gold per ton. However, indications from advance drilling and geological interpretation indicate that additional reserves will be found to prolong the life to a minimum of 15 years.

At the end of 1984 proven and probable reserves were estimated to contain 1.35 million ounces of gold, enough for 8-9 years' production at current mining rate.

Operating Costs

Cash breakeven costs in 1984, excluding financing costs and exploration were $US 198 per ounce of gold recovered, down from $263 in 1983.

History

Lupin went into operation in October, 1982 at an initial milling rate of 1,000 TPD. In late 1983 the capacity was increased to 1,600 TPD.

Geology and Mineralogy

Gold occurs in an iron formation within an amphibolite lens that is highly folded. About 10-15% of the ore is quartz in stringers and veins up to 5 feet in width.

The ore contains about 10% pyrrhotite but locally it may run as high as 30%, a lesser amount of arsenopyrite, and minor pyrite. The sulphides occur close to the quartz or between the quartz and iron formation.

Gold occurs predominantly in the native form, within the pyrrhotite and along the grain boundaries of the arsenopyrite. Grain size of the gold is 50 microns or less. It is estimated that 30% of the gold is associated with arsenopyrite and the balance with pyrrhotite. Arsenopyrite and pyrite grains are "gold poor."

Mining Methods

Most of the ore is mined by sub-level, retreat, long-hole stoping method. At present the mine is in permafrost which is estimated to extend to a depth of 1,500 feet. A brine solution is used for drilling to prevent the drilling fluid from freezing.

Annual Tonnage Milled/Ore Grade

For the full year 1984 approximately 542,000 tons of ore were milled with an average grade of 0.35 ounces per ton.

Gold Recovery %

Typical gold recovery is 95%.

Gold Production

In 1984 gold production was 181,534 ounces.

Milling Process

The process is conventional straight cyanidation. Due to the presence of pyrrhotite in the ore, preaeration is necessary to conserve cyanide and lime in leaching. The circuit includes the following:

Crushing - 4 in. ore from underground in two-stage cone crushing to 3/4 in.

Grinding in a rod mill followed by two ball mills in parallel to 80-84% - 200 mesh.

Thickening of ground product (cyclone overflow).

Preaeration with lime (3 tanks).

Leaching with lime, sodium cyanide and lead nitrate for 32 hr. at 45% solids.

Two-stage vacuum drum filtration.

Gold recovered by Merrill-Crowe process.

Filter cake repulped and pumped to tailings.

Natural degredation of cyanide in tailings pond.

Almost no instrumentation was incorporated in the plant although all the equipment was new. Lupin intends to add some, for example, a gamma gauge on the thickener underflow.

Types of Refractory Ore

Unlike most gold ores containing arsenopyrite the Lupin ore is not refractory because gold occurs on the crystal boundaries of the arsenpyrite but not within the grains.

Problems in Processing the Ore

Although the ore is not refractory, it is necessary to prearate it prior to cyanidation. This was originally carried out at cyclone overflow density (33-36% solids) but difficulty was experienced with sanding out of the air lift agitators used for preaeration. Late in 1984 Lupin planned to change the circuit to preaerate after, rather than before thickening since no difficulty has been experienced with the same type of agitator at higher density (45% solids) in leaching.

Lupin keeps the lime low, around pH 10.3 in leaching to minimize dissolution of arsenopyrite. There has been some concern with evolution of hydrogen cyanide at this pH and in 1984 the leach tanks were being covered and vented.

Comments

Lupin believes that lead nitrate is necessary in leaching to maximize gold recovery. Typical addition is 0.45 lb. per ton of ore.

Original laboratory work on Lupin ore showed that 97 to 99% of the gold could be recovered by flotation at a coarse grind, in the range 50-60% minus 200 mesh. By regrinding the concentrates to 85% minus 400 mesh leach extractions of 90.6% and 95.0% were obtained on two samples that were tested. For some reason no additional tests were run to optimize conditions for cyaniding the concentrate.

Use of salt for drilling in the mine results in some salt in the milling process. This has not caused any noticeable corrosion or process problems in the mill. Some corrosion was experienced with underground equipment but a switch was made from calcium to sodium chloride and this greatly alleviated the problem.

Lupin may be the only gold mill in Canada to use pressure leaf filters for clarification of pregnant solution. These units are performing well on automatic control.

Lupin has tested carbon-in-pulp on their leach slurry with very good results. If the mill is expanded Lupin may incorporate CIP in the new circuit.

Outlook

With high grade ore, efficient mining and milling methods and well-regarded management Lupin can probably look forward to many years of operation. An expansion in the rate of production appears likely.

References

1. Edwards, I. C. Report to CANMET on Visits to Mines in the Northwest Territories, September, 1984.

2. The Northern Miner, January 31, 1985, Pp. 1, 6

GIANT YELLOWKNIFE MINE

Location

The Giant Mine is located on Yellowknife Bay, 6 km north of Yellowknife, N. W. T.

Ownership

Falconbridge Limited, with 19.2% of Giant shares effectively controls the company.

Economic Metals

Gold, some silver, and crude arsenic trioxide.

Ore Reserves

Proven and probable reserves at Giant as of December 31, 1984 were estimated at 1,057,000 tons grading 0.24 ounces of gold per ton, down slightly from a year earlier. Reserves include allowances for mine dilution.

Operating Costs

Mine operating costs were $79.22 per ton of ore, equivalent to C $402 (US $311) per ounce of gold produced in 1984.

History

Milling commenced at Giant in 1948 at a rate of 250 TPD. An Edwards type roaster went into operation early the following year.

Most of the gold produced at Giant has come from the basic process of flotation, roasting of concentrate, and cyanidation of roaster calcine and dust. During early operation some gold was recovered by jigging and amalgamation. Between 1957 and 1967 a small portion of the gold was extracted by cyaniding the flotation tailings. Since 1967 gold has been extracted exclusively from cyanidation of roaster products.

The present two-stage fluid bed roaster and gas cleaning plant went into operation in 1958 and have operated with only minor modifications since that time.

Geology and Mineralogy

Gold occurs throughout sericite schists cutting Precambrian rocks. The ore is characterized by 40 - 80% quartz-carbonate and approximately 8% pyrite-arsenopyrite. Significant amounts of stibnite and antimony sulphosalts are also present. At least 70% of the gold occurs as sub-microscopic particles intimately associated with arsenopyrite or in solid solution with it.

Mining Methods

Open pits provided about 40% of the mill feed and underground mining the remainder.

Annual Tonnage Milled/Ore Grade

In 1984, 325,548 tons of ore were milled at an average grade of 0.23 ounces per ton.

Gold Recovery, %

Gold recovery averaged 86.0% in 1984.

Gold Production

Gold recovered in 1984 amounted to 64,066 ounces.

Arsenic trioxide production was not reported but shipments in 1984 were 1,234 tons.

Milling Process

Mill capacity is 1,250 tons per day but has operated recently only 5 days per week to conserve ore reserves.

Basically the process consists of the following:

Three-stage crushing to 3/8 in.

Single-stage ball milling to 55% - 200 mesh.

Three-stage flotation and regrinding of primary flotation tailings to 75% - 200 mesh.

Flotation tailings to mine backfill and tailings pond.

Two-stage roasting of flotation concentrate.

Roaster gas cleaning by electrostatic precipitation and, after cooling, bag filtration to recover arsenic trioxide.

Regrinding of roaster calcine to minus 325 mesh.

Cyanidation of calcine, followed by gold recovery by Merrill-Crowe process.

Cyanidation of Cottrell dust followed by carbon adsorption of gold.

Carbon shipped to a smelter to recover gold.

Arsenic from baghouse shipped to the USA.

Tailings effluent treated to break down cyanide compounds and remove arsenic.

Gold scavenged from tailings effluent by carbon adsorption.

Types of Refractory Ore

The ore is refactory because most of the gold is intimately associated with arsenopyrite.

Percent of Gold in Refractory Ore

At least 70% of the gold occurs in arsenopyrite in a form that requires roasting or chemical disintegration to release it.

Percent of Refractory Ores to Total Ores

There may be some zones that contain ore that is not refractory but it is not practical to separate this material for separate processing.

Problems Related to Presence of Refractory Ore

In the past Giant has experienced environmental problems in-plant, in the atmosphere and in the tailings effluent, due to arsenic compunds. These problems have been effectively overcome.

For many years Giant produced arsenic that was not marketable. A large tonnage of this dust was stored in sealed stopes underground, at considerbale expense.

Outlook

Giant is a marginal producer at the current value of gold and its future is uncertain.

References

1. Giant Yellowknife Mines Limited Annual Report, 1984.
2. Edwards, I. C. Report to CANMET on Visits to Gold Mines in the Northwest Territories, September 1984.

SALMITA MINE

Location

The mine is located on the shore of Matthews Lake, 150 miles northeast of Yellowknife, N.W.T. The ore is treated in the former Tundra Mill, some 6 Km southeast of the mine.

Ownership

Salmita is a division of Giant Yellowknife Mines Ltd.

Economic Metals

Gold, with minor amount of silver.

Ore Reserves

Proven and probable reserves at year-end 1984 were 88,000 tons containing 0.85 oz/ton gold, down to the 6th level (950 feet).

Operating Costs

Operating costs in 1984 were $211.10 per ton and $288.51 (US $222.81) per ounce.

History

Gold was discovered on the property in 1939 and the property staked in 1945. Giant optioned the property in 1975. An underground exploration program and feasibility study in 1981 led to a production decision. Milling commenced in August, 1983. The ore is processed in the Tundra mill which was extensively rehabilitated in 1983 prior to production. Operations have continued steadily since 1983.

Geology and Mineralogy

Salmita is one of several gold showings associated regionally with the contact zone between younger, finely laminated argillaceous metasediments in the east and older intermediate to felsic volcanics in the west. Locally, the Salmita deposit is contained within a persistent 1.7m wide zone of intense epithermal silicification with free milling gold associated with some scheelite and minor sulphides (arsenopyrite, pyrite and trace amounts of pyrrhotite, galena and sphalerite).

Mining Methods

Shrinkage stoping is used due to the narrow veins, averaging 1.5m wide. Ore is hauled up a 13% decline in the footwall of the orebody.

Annual Tonnage Milled/Ore Grade

Salmita output in 1984 totalled 60,702 tons at a grade of 0.764 ounces gold per ton. Grade improved from 0.63 ounce in the first quarter to 0.93 ounce per ton in the final quarter.

Gold Recovery %

Recovery improved to more than 97% in the latter months of 1984.

Gold Production

In the full year 1984, gold production was 44,414 ounces.

Milling Process

Crushing to 3/8 inch in jaw and cone crushers.
Grinding to 70% - 200 mesh.
Straight cyanidation leach (36h).
Gold recovery by Merrill-Crowe process.
Precipitate smelted at Giant Yellowknife.

Due to the unusual hardness of the Salmita ore mill throughput was initially restricted to 125 TPD. With the installation of an 8-foot by 8-foot ball mill in June 1984, mill throughput was increased by 50 TPD.

Mill tailings are impounded in the original Tundra tailings area, which has had the berms raised. It is believed that natural degredation will effectively control cyanide when it becomes necessary to discard pond effluent.

Refractory Ore

The ore is not refractory when ground fine enough to liberate the gold.

Outlook

Ore reserves are very limited but Giant considers that the potential for replacing ore mined is quite good.

References

1. Western Miner, June 1984, Pp. 11, 12
2. Northern Miner, July 19, 1984, Pp. B-11
3. Can. Mining Journal, October, 1984, Pp. 11-13
4. Giant Yellowknife Mines Annual Report, 1984.
5. Personal communication, S. Fekete

SECTION 2

HEMLO AREA MINES

NORANDA'S GOLDEN GIANT MINE

Location

The mine is located about 30 miles north-west of White River in Northwestern Ontario.

Ownership

According to a 1982 option agreement, Noranda earns a 50% interest in 156 claims held by Golden Sceptre Mines Ltd. and Goliath Gold Mines Ltd. by placing the property into production.

Under a separate agreement with Teck Corp., Noranda has an option to earn a 50% interest in a quarter-claim property from the Teck-Corona joint venture.

Economic Metals

Primarily gold but some molybdenum may be produced.

Ore Reserves

Ore reserves are estimated at 20 million tonnes with an estimated grade of 0.28 ounces of gold per tonne.

The Teck-Corona quarter-claim has reserves estimated at 2.1 million tonnes with 0.33 ounces of gold per tonne.

Operating Costs

Noranda has projected costs of $130 per ounce of gold produced when the mine is at full capacity. Successful molybdenum production could result in a credit of $20 to $30 to the cost of producing gold. Noranda's cost could be one of the lowest in the world.

History

A shaft is being sunk to an eventual depth of 1,250 metres. Mill tune-up was scheduled for the first quarter of 1985 with full production

of 1,000 tonnes per day expected before October. Plans call for that capacity to be increased to 2,500 tonnes by early 1987, and 3,000 tonnes by mid-1988.

Geology and Mineralogy

The geology is similar to that of Lac Minerals and Teck-Corona, all three mines being part of one massive ore zone. There is some pyrite, arsenopyrite, molybdenite and stibnite in the ore. About half of the gold is closely associated with the arsenopyrite.

Mining Methods

Blast-hole stoping, a bulk mining method, will provide most of the mill feed.

Annual Production

The current plan is for production of 280,000 ounces of gold in 1987, rising to around 336,000 ounces per year when the mine reaches the 3,000 tonnes per day rate.

Gold recovery is expected to be around 95%.

Milling Process

The process is generally similar to that of the other two mines in the area except that Noranda is using conventional crushing and grinding, rather than semi-autogenous grinding, and the Noranda mill is recovering a gravity concentrate.

On surface the ore is crushed to 1/2 in. using two-stage cone crushing, and ground to 92%-200 mesh in three-stages of grinding. There is some relatively coarse gold in the ore which is recovered by jigging. Jig concentrates are reground and treated by intensive cyanidation.

The ground product from the main grinding circuit is treated by preaeration, cyanide leaching, carbon-in-pulp adsorption, stripping,

electrowinning and smelting to produce bullion.

Leaching is carried out at relatively low pH, less than 10.7, due to the presence of arsenopyrite and stibnite in the ore.

Noranda has a flotation circuit in the mill to recover molybdenite from the cyanide tailings. It is not known yet whether this circuit will produce a marketable moly concentrate. Average molybdenum content of the ore is 1.5 grams per tonne.

Refractory Ore

The ore is not refractory but very fine grinding is necessary to liberate the gold and preaeration is necessary to reduce cyanide consumption and, possibly, to increase gold extraction in leaching.

Barite

The Hemlo ores contain appreciable amounts of barite. It is possible that this may be recovered as a by-product for oil drilling mud.

Outlook

With reserves of high grade ore sufficient for 20 years at projected production rates, Nordanda's Hemlo mine should be a major gold producer for many years.

References

1. Borland, James. "Deep blue corporate colors of Noranda put lasting stamp of bigness on Hemlo." Northern Miner, January 17, 1985, Pp. B4, B5.
2. The Northern Miner, January 17, 1985, P. C5.
3. "The Hemlo Mines - an update." CIM Bull. February, 1985, P.12.
4. Personal communication with K. Konigsmann.

LAC MINERALS HEMLO MINE

Location

Lac's Hemlo mine is located 30 miles north-west of White River in North-Western Ontario.

Ownership

The mine is owned and operated by Lac Minerals Ltd.

Economic Metals

Gold

Ore Reserves

Reserves at Lac are estimated to be 52 million tons with an average grade of 0.18 ounces per ton, the largest reserves in the Hemlo camp.

Operating Costs

Projected operating costs for mining, milling and administration are $36 per ton, equivalent to $175-$200 per ounce of gold produced.

History

The mine is slated to go into production in April 1986, or possibly earlier at an initial rate of 3,300 tons per day with expansion to 6,600 tons per day by 1989.

Geology and Mineralogy

Gold deposits at Hemlo occur in mineralized beds of metasediments related to volcanic activity. The geology is complex and the origin and

formation of the gold is not fully understood.

Gold occurs in veins with pyrite and other sulphides. Unique to Canada, the mineralized zones also contain barite, which may constitute more than 30% of the beds in places. Stibnite, realgar and molybdenite also occur in the ore zones.

Generally, in the Hemlo area gold occurs as regular, homogeneous distribution of finely disseminated grains or free gold throughout the ore zones, and grade tends to increase with depth.

The ore deposits have simple geometrics, sheet-like and dip at 65^{o}-70^{o}. Deposits are large with consistent widths of 10-125 ft., high vertical dimensions and strike lengths of approximately 6,000 ft. These characteristics make for bulk mining methods, low costs and ease of planning flexible production.

Mining Methods

Less than one million tons will be mined by open pit and the balance by underground bulk mining methods. A mixture of waste rock, sand and cement will fill the stopes after the ore is withdrawn.

Annual Production

At the initial milling rate of 3,300 tons per day, and assuming a mill feed grade of 0.18 ounces per ton, 92% operating time and 90% extraction, gold production will amount to 180,000 ounces per year. This production will double when the operation is expanded to 6,600 tons per day by 1989.

Milling Process

The new mill will employ "state of the art" technology including the following unit processes:

Semi-autogenous primary grinding.
Ball-milling and two-stage classification to 92%-200 mesh.
Thickening.
Preaeration (2 tanks) for 16 hr. with lime.
Cyanide leaching in 5 tanks (40 hr.).
Gold adsorption by carbon-in-pulp in 6 stages.
Pressure stripping at 130°C. and 360 kpa.
Electrowinning and smelting of gold.
Carbon regeneration at 600°C.
Solid wastes pumped to tailings pond, with effluent recycle.
Possible effluent treatment in the future (no discharge for first two years).

Refractory Ores

No refractory ore has been reported.

Outlook

With huge reserves of fairly high grade ore, Lac's Hemlo mine is likely to become Canada's largest gold mine, both in terms of tonnage rate and gold production.

References

1. Hope, George H. "Lac Minerals Development of Hemlo" 17th CMP Conference, January 1985.
2. Wahl, W. G. "Hemlo gold occurrences not common type," Northern Miner, July 14, 1984, P. 19.
3. Northern Miner, January 17, 1985, P. C2.
4. CIM Bull., February 1985, P. 12.

TECK-CORONA MINE

Location

The Teck-Corona mine is located between Marathon and White River in the Hemlo area of Northwestern Ontario.

Ownership

The Teck-Corona property is owned by Teck Corporation Limited and Corona Resources Limited. Under a development contract, Teck will earn a 55% interest in the project upon bringing it into production.

Economic Metals

Gold

Ore Reserves

Reserves are estimated at 8.4 million tons with an average grade of 0.36 ounces per ton of gold per tonne. In addition, Teck-Corona and Noranda will share equally in profits from a quarter-claim which contains an estimated 2.1 million tons grading 0.33 ounces per tonne.

Operating Costs

Projected operating costs when the mine is in full production should reach about $US 110-120 per ounce of gold produced.

History

The mine is preparing to go into production with the mill probably operating by April 1985. A shaft is being sunk to an ultimate depth of 3,800 ft.

Geology and Mineralogy

The Hemlo deposit is "stratabound," occurring in a band of tuffaceous sedimentary rocks. The gold, as well as some molybdenite and barite was probably deposited as a chemical sediment at the time the tuffs were depositing.

The deposit consists of an inclined, tabular sheet of mineralization that is relatively uniform in grade across a length of 6,500 ft. along strike, and of at least 4,000 ft. down dip.

Mining Methods

Mining methods to be employed will be sub-level open stoping with delayed backfill, and mechanized cut and fill stoping.

Annual Production

The mill is rated at 1,100 tons per day and when in full production should produce at least 125,000 ounces of gold annually.

Milling Process

The process is almost identical to that at Lac Minerals, consisting of semi-autogenous and ball mill grinding, pre-aeration, cyanide leaching, carbon-in-pulp adsorption, stripping, electrowinning and smelting. Tailings will be cycloned in 3 stages to produce backfill.

Tailings will be impounded behind an impervious dam 1.5 miles from the minesite. Decant water from the tailings pond will be recycled to the mill. Effluent from the pond will be treated to meet Ontario Department of the Environment requirements.

Gold recovery is expected to be 95%.

Refractory Ores

The ore is not refractory but, because of the presence of pyrite it is preaerated prior to leaching.

Outlook

With high grade ore, low operating costs and 20 years ore in sight at the initial production rate, Teck-Corona is likely to be a major gold producer in Canada for many years to come.

References

1. Teck Corporation Annual Report, 1984.
2. Sibbalt, C.V. "Just so much rock, then metallurgists take over." Northern Miner, January 17, 1985, P. C2.
3. "Looking for low costs" Northern Miner, January 17, 1985, P. C20.

SECTION 3

PROSPECTIVE GOLD MINES

Agassiz Resources - Sherritt Gordon

Sherritt has an option on Agassiz Resources gold property adjoining the Lynn Lake mine in Northern Manitoba. Sherritt has spent $9 million exploring the property and is committed to spend an additional $7 million to deepen the shaft to the 1,650 ft. level and explore the lower levels.

Probable, possible and inferred reserves to the 1,600 ft. level were estimated to be 3.5 million tons averaging 0.17 ounces per ton, in late 1984. Minable reserves above the 1,200 ft. level, using cut-and-file mining methods are 1.8 million tons averaging 0.2 ounces per ton gold.

Sherritt anticipates operating at a rate of 1,000 tons per day if minable reserves can be increased to project 8-10 years mine life. The ore could be milled in the Lynn Lake mill by converting part of it to treat gold ore.

At least nine gold-bearing veins containing sulphides (sphalerite, galena, chalcopyrite, pyrrhotite and pyrite) have been discovered in a small amphibolite body within mafic volcanics.

Published reports on the Agassiz development make no reference to any processing difficulties with the ore.

References

1. Mineral Bulletin MR 198, May, 1983, P. 191.
2. Northern Miner, Nov. 8, 1984, Pp. 1, 2; Dec. 20, 1984, Pp. 1, 2; April 4, 1985, P. 3.

AGIP - Erickson

AGIP Canada's Mount Skukum gold deposit is under development by Erickson Gold Mines. The deposit is located 40 miles southwest of Whitehorse, Y. T.

Erickson is currently driving a crosscut into the main ore zone where 165,000 tons of reserves have been blocked out grading 0.73 ounces of gold and 0.63 ounces of silver per ton.

The deposit is described as a caldera, composed predominantly of inter-basic volcanics with flows and tuffs. Gold values occur in quartz carbonate with hardly any sulphides.

Preliminary metallurgical testing was very favourable. A nominal production rate of 200 tons per day is being considered. Erickson is considering moving the Plaza Mining mill, which it owns, from Cassiar, B.C. to the AGIP minesite.

Reference

1. Northern Miner, Nov. 1, 1984, P. 1.

Blackdome's Clinton Mine

A production decision is expected soon on Blackdome Exploration's Clinton property 40 miles west of Clinton in southern B. C. The property has been under development since 1979.

Reserves are estimated to be 195,000 tons averaging 0.78 ounces of gold and 3.2 ounces of silver per ton. Cut-and-fill dilution is expected to be 25%, due to poor ground conditions.

The ore is reported to be classic caldera-related epithermal mineralization similar to producing gold and silver camps in Mexico and the U.S. southwest.

The metallurgy of the ore is fairly simple - some 60% of the gold should be recovered by jigs. There are no deleterious elements in the ore and the sulphide content is low. A milling rate of 200 TPD has been recommended by consultants.

References

1. Northern Miner, Nov. 29, 1984, Pp. 1, 14; March 7, 1985, P. 1.

Casa Berardi (Inco/Golden Knight)

Inco Ltd. has discovered two gold deposits in the Abitibi greenstone belt some 150 km. north of Noranda, Quebec and 50 km. southeast of the Detour Lake mine. Inco considers these deposits to be amongst the most important undeveloped gold prospects in Canada.

Claims cover a strike length of 40 km. extending from Bradette Township, Ontario through five townships in Quebec. Inco and Golden Knight Resources own 60% and 40% respectively of 875 claims, and together own 60% of 17 Domex claims.

Gold was discovered by Inco in April, 1981 and ore reserves have expanded steadily since that time. In the original "Main" Golden Pond deposit drill-indicated reserves are estimated to be 3 million tons grading, 0.22 ounces of gold per ton. In the East Zone, some 2,000 m east of the Main Zone reserves will probably contain a similar tonnage with an average grade of 0.29 ounces per ton.

The geology in both zones is similar and complex. Host rocks include a variety of volcanic and sedimentary rocks, including iron formations. Gold mineralization is found in all rocks and is related to a major fault zone.

Quartz veins, of varying thicknesses, and usually showing visible gold, are found in direct association with pyrite - arsenopyrite mineralization in adjacent sediments or volcaniclastics.

Metallurgical characteristics are similar in both zones, with gold mainly free, find grained, and easily recovered by cyanidation. Preliminary tests indicate recoveries of at least 90%.

References

1. Northern Miner, March 14, 1985, Pp. 1, 6.
2. Church, J.F. et al "Casa Berardi - A Case History of Successful Gold Exploration" PDA Convention, Toronto, March, 1985.

Cameron Lake

Echo Bay Mines has an option to develop Nuinsco Resources'/Lockwood Petroleum's properties in the Cameron Lake area, 50 miles southeast of Kenora, Ontario.

Ore reserves have been estimated at 1.5 to 2.0 million tons averaging 0.15 ounces of gold per ton or 900,000 tons averaging 0.195 ounces per ton.

The highest grade material is confined to siliceous pyritic breccia veins within a carbonate-sericite-pyrite alteration envelope. Veins are up to 30 ft. in width and 140 ft. in length. There is potential for an open pit operation.

Echo bay expects to spend one million dollars on the property in 1985.

References

1. Northern Miner, Sept. 13, 1984, P. 22; Feb. 14, 1985, Pp. 1, 2; March 14, 1984, P. 22.

Duport Mine

The Duport gold property is located approximately 25 miles southwest of Kenora, Ontario. The headframe is located on Cameron Island in Shoal Lake. The property is being developed by Consolidated Professor Mines Ltd. and joint venture partner Union Carbide Exploration Corp.

Originally discovered in 1896, the property has been explored several times and a limited amount of gold and silver produced up to 1936. In recent years Consolidated Professor and its joint partners have carried out extensive surface and underground exploration and two mineralized zones have been outlined.

Ore reserves in mid-1984 were estimated to be two million tons grading 0.30 ounces of gold per ton across an average width of 9.7 ft.

The two ore zones lie within a zone of intense deformation in intermediate to basic volcanic rocks. Gold is hosted by well-developed quartz- and sulphide-rich zones. It appears to have a strong association with arsenopyrite, both in solid solution and also as free gold encrusted along grain boundaries. Free gold is also found in lesser amounts associated with pyrrhotite, pyrite and chalcopyrite.

Development of the mine has been hampered somewhat by the fact that the mine is located on a small island in a prime resort area and it will be necessary to transport the ore to a mill on the mainland, across the water or by means of a tunnel to the mainland. There may also be some metallurgical problems with the ore, depending on the proportion of gold in solid solution in arsenopyrite. Gold production costs have been estimated to be in the $ US 275 per ounce range.

References

1. EM & R Mineral Bulletin MR 198, P. 158.
2. Northern Miner, June 28, 1984, P. 12; July 19, 1984, Pp. 1, 2, 3.

Davidson Tisdale Property

Getty Mines Ltd., in a joint venture with Davidson Tisdale Mines, is exploring the latter's property in Tisdale township, near Timmins, Ontario.

The property saw limited development from discovery in 1909 to its closing in 1920. A shaft on the property extends to a depth of 812 ft. and five levels have been developed.

Drilling along 2,600 ft. of strike to the 650 ft. level has outlined 824,000 tons of ore with an average grade of 0.36 ounces of gold per ton (uncut) over a true width of 10 ft.

Another small deposit contains 171,000 tons with an average grade of 0.33 ounces per ton.

Quartz-carbonate tourmaline bearing veins host the gold mineralization. The quartz veining and wall rock alteration are similar to that found at the Hollinger and McIntyre mines approximately 5 km. to the southwest.

If underground development confirms the drill-indicated reserves the mine will probably go into production. It is conceivable that the ore might be treated at one of Pamour's mills in the area.

References

1. Northern Miner, Oct. 11, 1984, P. 1; Dec. 20, 1984, P. 18.

Hoyle Pond - Kidd Creek

The Hoyle Pond gold deposit is located beneath Kidd Creek Mines' tailings pond near Timmins, Ontario. In 1984, Kidd Creek completed the first phase of an underground development program, including completion of a 750 m. ramp to the ore zone and sampling to establish the continuity and grade of the main vein. A large bulk sample was mined from the 120 m. level and treated in Asarco's Aquarius mill. The bulk sample returned an average grade of 0.55 ounces of gold per ton.

The ore is free milling and approximately 97% of the gold in the bulk sample was recovered by jigging and flotation. The concentrate was returned to Kidd Creek's smelter.

Kidd Creek have budgeted to spend $16 million over the next two years to develop the mine and build a 300 TPD mill.

References

1. Northern Miner, June 14, 1984.
2. Kidd Creek Mines Annual Report, 1984.
3. Personal communication, C. Chapman.

APPENDIX

TRIP NOTES AND ADDITIONAL INFORMATION

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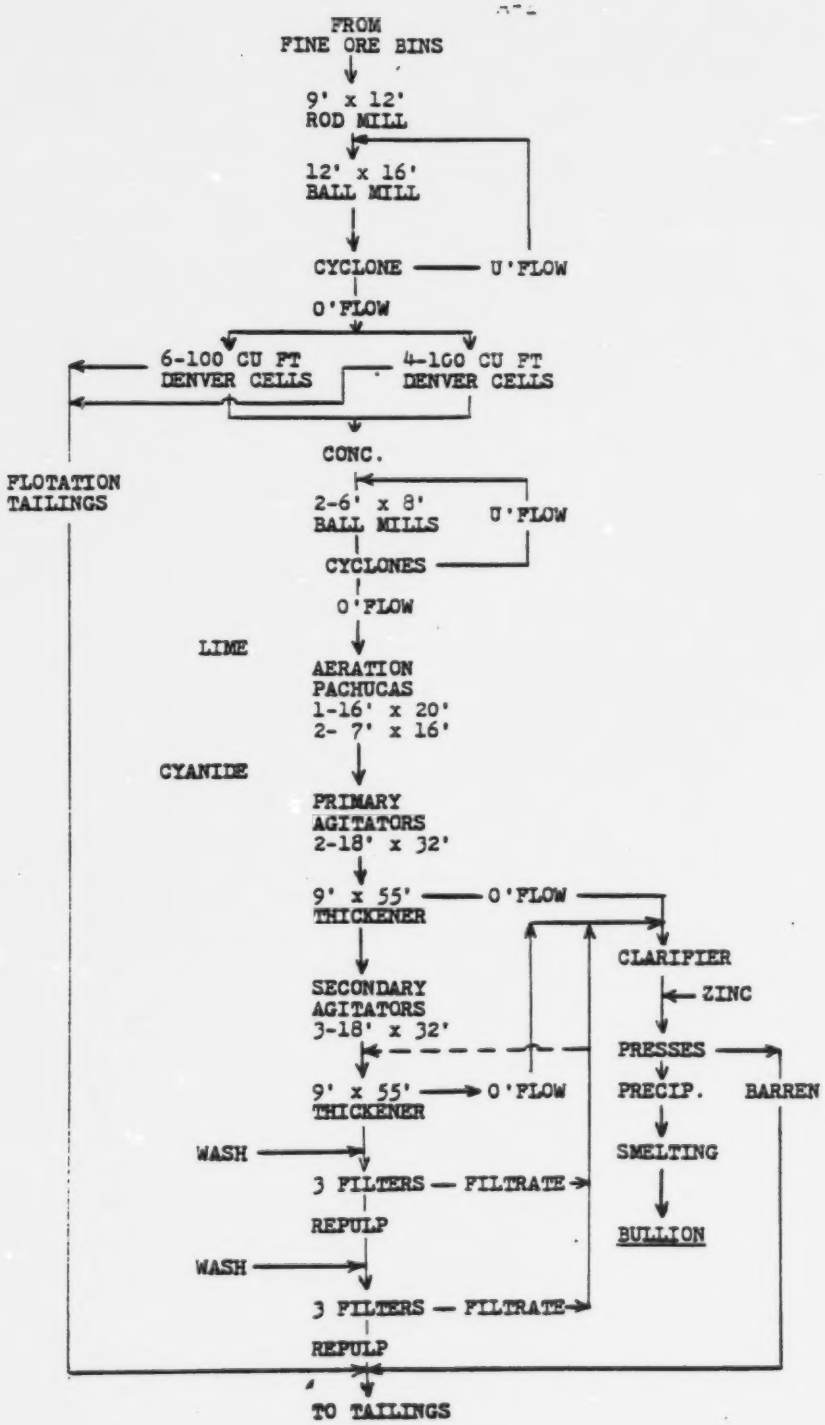


I.C.E.
March 1985

FLOWSHEET OF AGNICO-EAGLE GOLD MILL

Bachelor Lake gold mines on stream debt free

By TERRY HAWKINS

Bachelor Lake Gold Mines Inc is a newly incorporated (March 1980), Quebec based company with rare total equity financing. A debt free picture is particularly attractive in these times of record high interest rates.

Financing was provided by a public offering, in June of 1980, of 3.0 million Bachelor Lake Gold Mines treasury shares for $10.5 million, yielding $9.7 million after financing costs.

Project cost overruns were provided for in these original arrangements, including the right to borrow $500,000 from Coniagas Mines Limited. Coniagas is a substantial shareholder of Quebec Sturgeon River Mines which owns 55 per cent of Bachelor Lake Gold Mines' outstanding issued shares. Bachelor Lake has an elective right to borrow a further sum of up to $1.0 million from a Canadian chartered bank together with the facility to obtain additional funds of up to $800,000 for working capital.

Total construction costs for all surface facilities, excluding the trailer camp amounted to approximately $9.25 million – a pretty reasonable price considering the headaches involved with developing a northern mine and surface facility in winter. The logistics of the project are staggering.

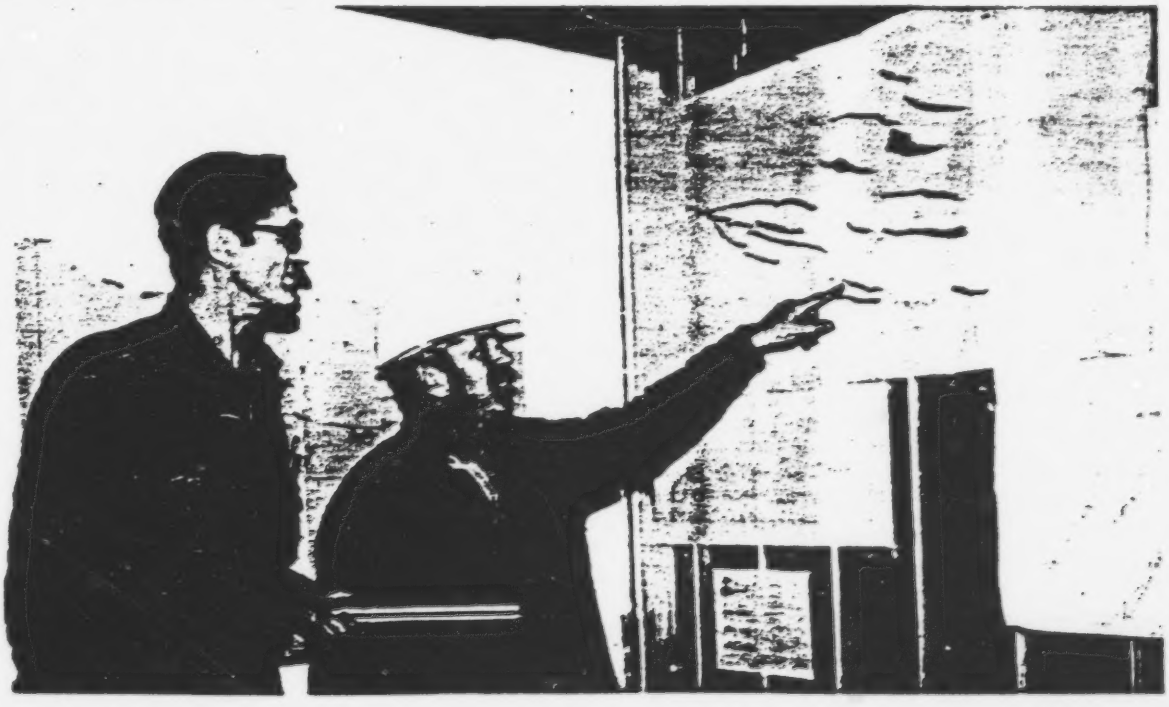
Two years of planning and design went into the Bachelor Lake gold mine project before work on site began in July 1980. This design work was done by Leslie Engineering Ltd.

Mining will be done by the mining contractor, R.F. Min-Con Ltd. The contractor is responsible for everything from the collar down including all underground supervision, labor and equipment with the exception of hoist, skips, rock breakers and mine pumps. All development and stope preparation, after the mine start-up and underground maintenance, is in the mine contractor's sphere.

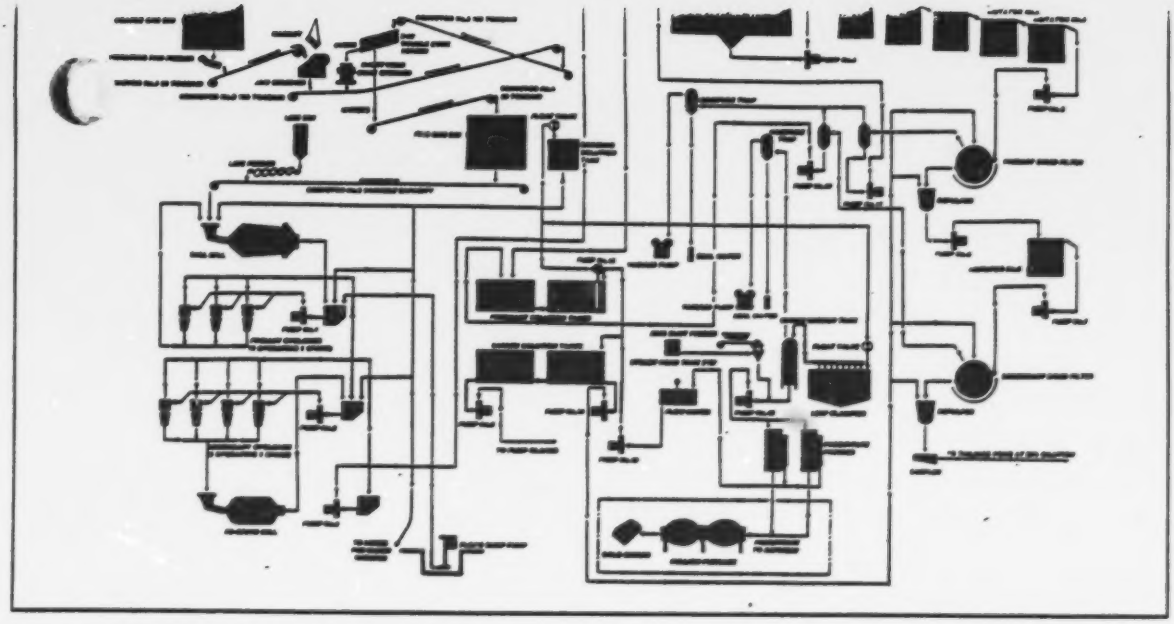
Power is supplied by Quebec Hydro. Bachelor Lake Gold Mines provided funds to construct a three mile power line connecting the mine's 44 kV substation to Quebec Hydro at the village of Desmaraisville. Costs of the power line will be refunded to the company based on an annual rebate of 25 per cent of the total power bill.

Mineralized zone in veins

The Bachelor Lake property is underlain largely by a series of interbedded agglomerates and tuffs of basic to intermediate composition with minor felsic



Yvan Tanguay, chief surveyor and Jean Arsenault, mine manager examine the Bachelor Lake gold mine deposits



Flowsheet of Bachelor Lake's 500 tpd gold cyanidation plant

volcanics. These rocks strike northeast on the property and dip steeply to the northwest.

The mineralized zone occurs in a cross-cutting fracture system striking approximately N 70° W and dipping about 55° SW. It consists of multiple quartz veins in a silicified and mylonitized reddish rock. The quartz consists of thin white threads and stringers, while hematite alteration provides the reddish tinge to the zone. On both sides of the mineralized zone the wall rock is highly silicified and contains albite, carbonate, white mica and chlorite along with very fine hematite.

The orezone has been partially developed on seven levels to a depth of 1,075 ft below surface. The southeast limit of development on most levels is the O'Brien granite stock, although it is known that the zone strikes through the granite and outcrops beyond the granite to the southeast.

From underground development done between 1973 and 1975 the company's geologists estimate ore reserves of 970,000 tons grading 0.195 oz Au/t (allowing 10 per cent dilution at 0.020 oz Au/t). Of this total 90 per cent is in the proven and probable category. Surface and underground diamond drilling has established the presence of ore below the current bottom level.

Stockpile milled first

Ore reserves are sufficient for five years production at the proposed milling rate of 500 tpd. In addition indications of ore both laterally and down dip should insure continuation of milling operations beyond the five-year period.

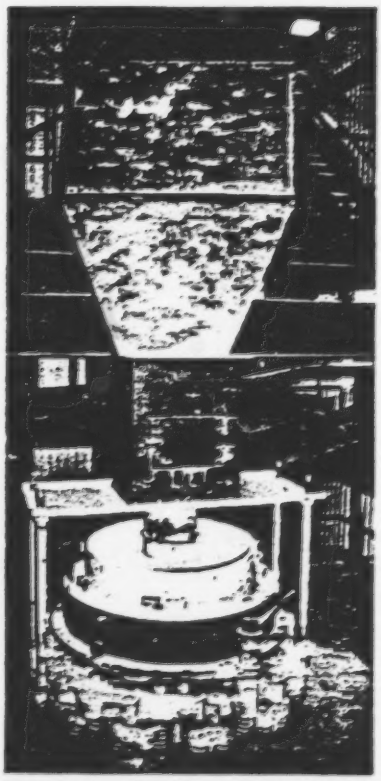
Initial mill feed will come from underground development and the surface stockpile. The stockpile totals approximately 17,000 tons of ore obtained from the original development program.

The surface stockpile is fed into the shaft skips at the mine collar by a 944 Pay Loader. The skips in turn dump into the coarse ore bin. This work is intermittent depending on the tonnage of ore or waste being hoisted from underground.

The 7.0 ft double drum Nordborg hoist is powered by a 500 hp electric motor. Hoisting speed is 1250 fpm. The shaft is equipped with a skip-cage combination with skip capacity rated at 3.5 tons. The loading pocket is situated below the 1050 ft level and mill feed will be hoisted from here during the first years of production. Capacity of the hoist is sufficient to carry the operations below the 2000 ft level.

The skip dumps the ore into a 500 t live load coarse ore bin. From the coarse ore bin the ore is fed to the 30 in. No. 1 conveyor, capacity 50 tph, by a vibrating pan feeder. This conveyor feeds a Kue-Ken 36 x 24 in. primary jaw crusher. A Dings magnet is installed at the head of the conveyor. This will remove any scrap metal that might damage the crusher.

The 100 ft x 24 in. No. 2 conveyor (capacity 150 tph) carries the ore to the transfer house, where it is discharged on to the 24 in. No. 3 conveyor (capacity 150 tph). This conveyor feeds the ore to a 3.0 x 6.0 ft Dillon double deck screen with 7/16 in. mesh opening.



Symons short head cone crusher (4¼ ft)

Anything oversize is fed into the 4¼ ft Symons short head cone crusher. The screen undersize feeds directly to the fine ore bin via No. 4 conveyor (capacity 50 tph). The fine ore bin is 32 ft in diameter and 30 ft high with a live capacity of 1200 tons. This is sufficient for more than two

Bachelor Lake were supplied and installed by Witby Welding Ltd.

To provide an accurate measurement of the tonnage produced by the mining contractor a Ramsey Totalizer, model 40-20, has been installed near the top of No. 4 conveyor. This weightometer measures weight of rock fed to the fine ore bin. The mining contractor will be paid on this tonnage, adjusted for moisture.

Ore is fed to the primary ball mill by the ore as it is carried along to maintain the desired pH level. The lime is added from a 36 ton storage bin via a screw feeder. It is estimated that about 2.0 lb of lime per ton will be needed.

The primary ball mill is a 500 hp 10.5 ft x 10 ft x 9.0 ft Tricone ball mill, with hardened Wabi manganese steel liners. Initial charge uses four sizes of balls – 1½ in., 2½ in., 3½ in. and 4 in.

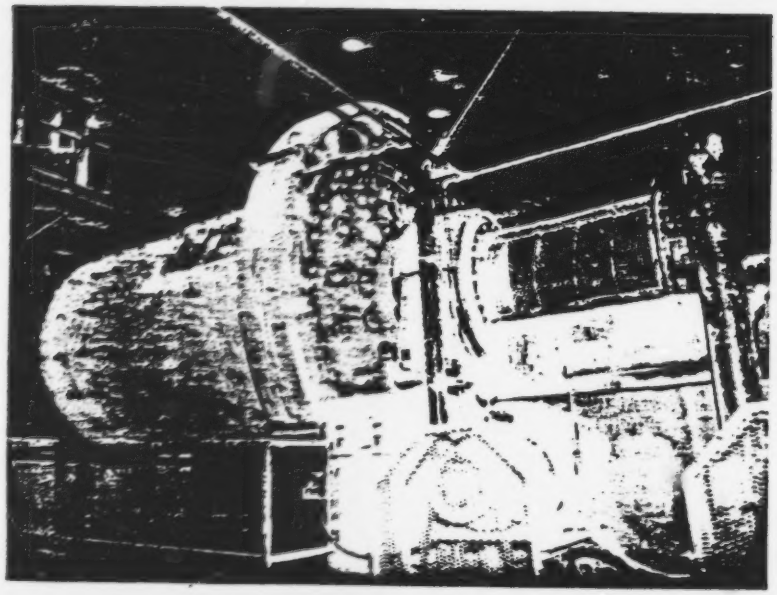
During grinding, sodium cyanide is added and the outlet feed (a slurry consisting of water, lime, muck and sodium cyanide) is pumped from the pump box to the 15 in. primary cyclones. Two are operating, one is held in reserve.

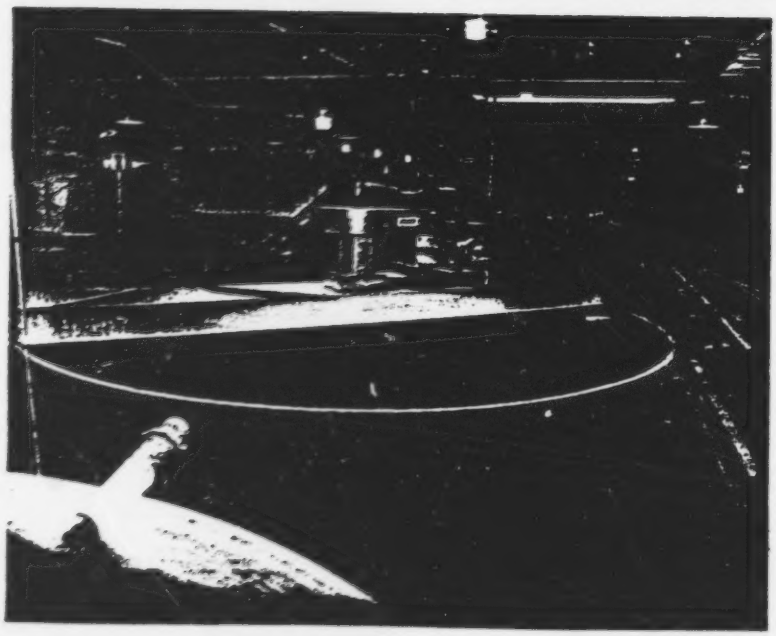
to the 10 in. secondary cyclones, where three are operating and one is held in reserve.

The coarse feed from the secondary cyclones is pumped to the 150 hp, 6.0 x 10 ft Dominion regrind mill, also lined with hardened Wabi manganese steel liners. The discharge from this mill is pumped back through the secondary cyclones. The fines (90 per cent minus 200 mesh) from the secondary cyclones are pumped to the thickener.

The thickener is a 40 ft diameter double tray unit. Its function in the circuit is to separate the easily dissolved gold from the slurry. About 40 per cent of the gold is in the liquor from the overflow and is pumped to one of the two 20 ft x 12½ ft pregnant solution tanks. The residue or pulp is pumped through the agitators.

Dominion regrind mill (150 hp, 6.0 ft x 10 ft). Freshly tightened bolts holding the new hardened Wabi manganese steel liners in place are clearly visable





Pulp treatment

In the thickener process, the heavier pulp sinks to the bottom of the thickener and is pumped to the first of six 22 ft x 22 ft Prochem agitators. The pulp is churned about by impeller as air is blown in through baffles around the edge of the tank.

Agitator time is about 36 to 40 hours from the pulp entering No. 1 agitator until it leaves No. 5. The principle is to encourage action by the cyanade and oxygen to dissolve the gold particles remaining in the pulp. The mixing process continues on the pulp as it flows from one agitator to another to get maximum benefit from the treatment. About 55 per cent recovery is expected at this stage.

From No. 5 agitator the pulp is pumped to the primary drum filter. Two Dorr-Oliver 12 ft diameter x 14 ft drum filters are being used, one primary and one secondary.

A clear filtrate is collected from the

Prochem agitator, No. 3 (22 ft diameter x 22 ft high)

Ball mill charge made up of 1½ in., 2½ in., 3½ in. and 4 in. steel balls

to the thickener and will enter the circuit again from there.

A rotating filter permits the mud s[illegible] on the exterior of the drum filter c[illegible] to be scraped off. This passes through the repulper and into No. 6 agitator.

The pulp from No. 6 agitator is pumped to the secondary drum filter. Water is added from one of the 20 x 21 ft barren solution tanks. Makeup water is supplied from the underground mine water or return water from the tailings pond. Residue from the filter goes directly to the tailings pond via a gravity pipeline.

Samples are taken all along the circuit to find out where the gold is at any given time. All assaying is done by the atomic absorption method.

Pregnant solution

The overflow from the thickener is pumped into pregnant solution tanks. Each of these tanks is 20 x 12½ ft. From the pregnant solution tanks the solution is pumped to the leaf clarifier.

The leaf clarifier is 15 ft x 9.0 ft with a sloping bottom, giving a maximum depth of 8.0 ft. The manifold is 6.0 in. in diameter, with 1.0 in. iron lubricated plug valves. The leaf clarifier removes the slime.

The pregnant solution from the clarifier is pumped into the 5.0 ft diameter x 12 ft high Crowe vacuum tank where the oxygen is removed. The clarifier overflow is pumped back into the pregnant solution. The solution is then fed to a press with zinc added to precipitate the gold in the Perrin gold press.

The barren solution from the press passes through the weir flowmeter to the barren solution tanks and back through the circuit.

The refinery is a 2000°F Wabi iron type reverberatory tilting furnace. Oil fired, it consists of a hood furnace and vent with a stack through the roof. Pots, bar molds, trolleys and mixing gear will soon contain the first gold bar from the Bachelor Lake gold mine.

Surface facilities

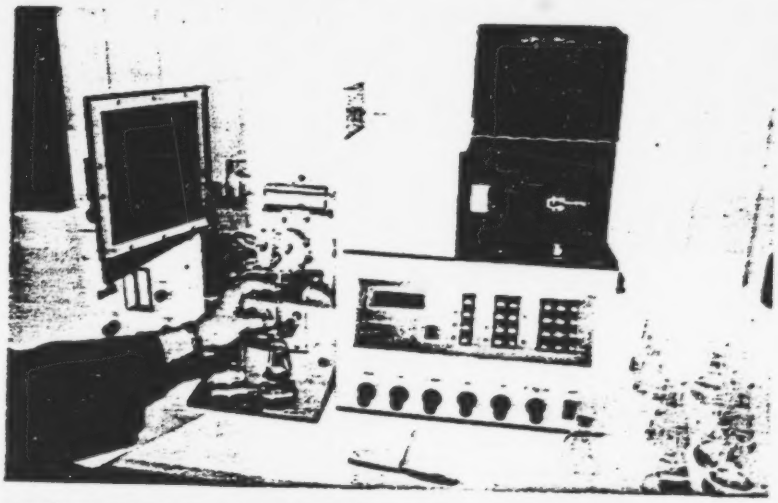
All of the materials used in constructing the Bachelor Lake mine were brought in by transport trailer truck. The buildings were partially prefabricated and are well insulated. Easy to heat with bright lighting, they are comfortable to work in even during the severe winter cold of northern Quebec. The headframe is of wood construction and 100 ft in height.

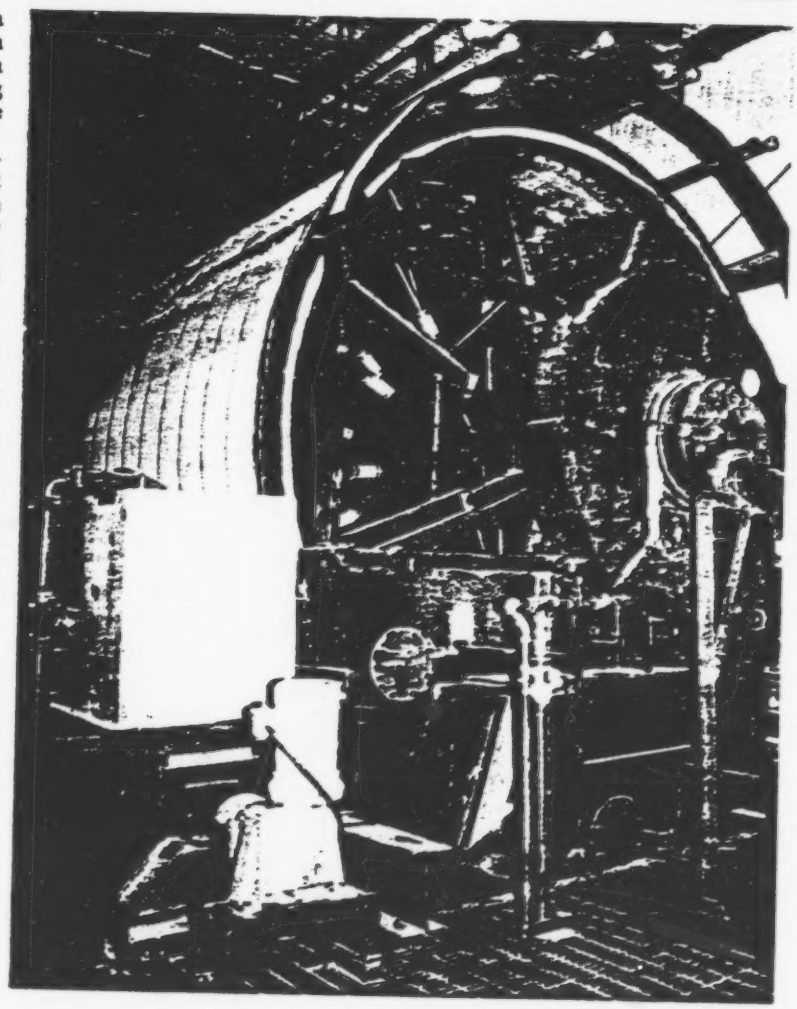
Dorr-Oliver 12 ft diameter x 14 ft drum filter

[illegible] water is supplied to the mill from a 95,000 gallon water tank erected on a nearby hill. An access road to the tailings pond has been completed. The initial tailings facility will be capable of storing up to 1.2

in the crushing plant and concentrator with two Richards and Wilcox overhead beam cranes.

Atomic absorption assay machine

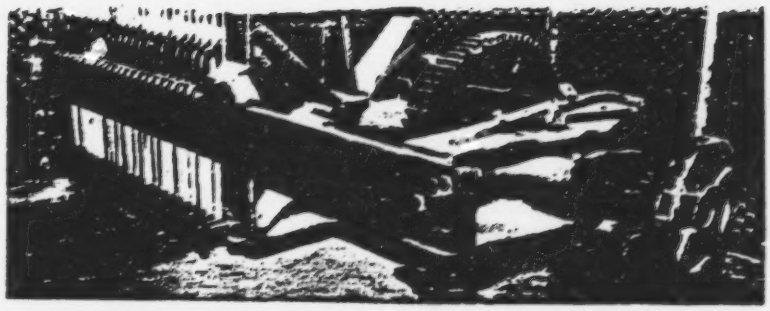


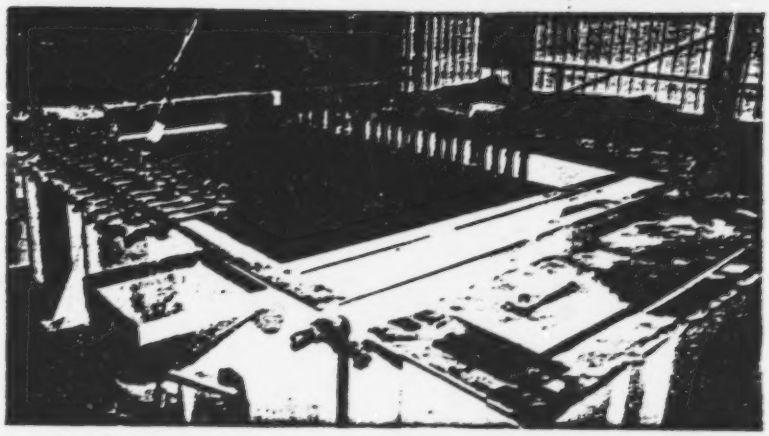


T[illegible]ler camp is made up of Atco trailers sufficient to comfortably accommodate 100 people. These units consist of five 20 man units, including cookery and three five man units. The camp site is about a mile from the mine. A dish satellite television receiver has been set up with wiring to each room to help entertain the crews during their off hours.

Safety at the Bachelor Lake mine has been an important consideration from the beginning. Any new construction is likely to have a few accidents, but this operation has been fortunate in this regard.

Dust is always a problem in crushing plants. At Bachelor Lake the screen and dust control system consists of a 30 hp 1800 rpm Carborundum TEFC dust collector with a centrifugal fan. Dust from the collector is returned to the circuit, as dust in a gold mill can be of considerable value.





Leaf clarifier, 15 ft x 9.0 ft with a sloping bottom, giving a maximum depth of 8.0 ft. The manifold is 6.0 in. in diameter, with 1.0 in. iron lubricated plug valves

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Notes of Visit to Camflo Mine, March 25, 1985

Met with O. Lafrance, Mill Superintendent.
Nominal mill capacity is 1,300 TPD.
Mill heads are currently 0.12-0.13 ounces per ton.

Grinding

Ball mills are lined with rubber. It is estimated that the shell liners will last for 8 years. Lifter life is 40 months (turned after 20 months).

Two-stage cycloning in the ball milling circuits is easily controlled. Grind is 88% - 200 mesh.

Thickening

The three thickeners are double-tray and are easily balanced.

No flocculant is used. Underflow density is 55-58%.

Thickeners have to be cleaned out twice a year due to build up on the rakes.

Leaching

Lime and cyanide are added to the ball mills; none of either is added later. A small amount of lead nitrate, about 15 lb./day is added to the rod mill to improve extraction. Tests showed that, when added to the ball mill or leach agitators it was detrimental. Lead acts as a catalyst and was reported to be most effective at high pH.

Leaching is in three stages of two agitators each and the slurry is filtered after each stage. Total retention time is 65-70 hr. About 80% of the extraction takes place in the ball mills.

Leach tank densities are 55-58% in the first stage, 60-63% in the second stage, and 65% in the third stage. Agitators are the air-lift type. In addition to the air lift, air is added through eight pipes in each tank.

Cyanide concentration	0.75-0.90	lb./ton of solution (initial)
Lime concentration	0.75-1.0	lb./ton of solution (pH 11.45-11.6)

Agitators have to be cleaned out once a year and this is a big job.

Filtration

Vacuum drum filters are flood washed.

Cakes appeared to have high moisture.

Filter cloths are punched felt or polyester.

Felt may filter a little faster than polyester and is about 25% cheaper.

Cloths last from 2 to 3½ years.

Presses

Four are installed but only two are in operation at one time. Pregnant volume is 3,200 - 3,300 TPD. The pH must be kept above 10.5 in the presses to prevent formation of arsine.

Personnel

Total personnel in the mill, excluding assayers but including maintenance is 27 or 28. Limited assistance is required from the shop and is confined mainly to assistance in changing mill liners.

Reagent Consumptions

Typical consumptions in lb. per ton or ore are as follows:

Rods	1
Balls	2
Lime	3.4
Cyanide	0.34 (NaCN equiv.)
Lead nitrate	0.012
Zinc	0.04
Lead acetate	0.002
Cyquest	0.025 (to clarifier feed)

Tailings

Water is used as wash on the third filter. Most of the barren is recycled to the mill.

Filter cake is repulped with water to 30% solids and pumped to a large tailings pond where natural degredation of cyanide takes place.

General

Camflo tried atomic adsorption but found that it wasn't accurate enough.

A new bulk handling system for lime is being installed; previously it was handled in bags.

Gold losses in the tailing are higher in the -325 mesh than in coarser fractions and it is suspected that some gold may precipitate in the circuit.

Operating Costs

Current mill operating costs including supervision but excluding mine administration, is about $5.25 per ton of ore milled. Camflo has a reputation as an efficient low-cost operation.

Notes of Visit to Aiguebelle's Dest-Or Mill, March 26, 1985

Camil Prince is Mill Superintendent.

Refer to attached simplified flowsheet of the process.

The fine ore bin, thickener and leaching tanks are all located outdoors. The ore bin and leaching tanks are constructed of steel and the thickener of concrete. There has been no difficulty from freezing in any of these.

Feed to the semi-autogenous grinding mill is controlled from a gamma-ray weightometer. This instrument is working well and is considered accurate enough for the material balance.

Metal liners in the SAG mill have shown high wear rates, otherwise the mill is operating satisfactorily.

The thickener is set on a pad, with no tunnel. The underflow is withdrawn with a SRL pump inside the mill through a 60-foot pipe on the bottom of the thickener. The density is measured with a gamma gauge. Pumping rate is controlled by throttling the pump discharge.

Leaching tanks are agitated with Pro Chem mechanical agitators which have given no problems. The tanks are 30 feet high.

Agitation in the carbon-in-pulp tanks, which are 18 feet high, is provided by double prop Pro Chem units. There is 1 tonne of carbon in each tank.

Haycarb C carbon is being used. It is pre-attritioned in a small tank for 10 hours prior to use. Loss is about 5%.

Carbon is transferred from column to column with Wemco recessed impeller pumps.

Once per day 1 tonne of carbon is transferred to stripping. The transfer takes about 1 hr. The strip columns are used alternately. They are washed occasionally with nitric acid.

All the carbon is regenerated each cycle whether it needs it or not. A vertical unit manufactured by Custom Equipment Co. of Salt Lake City, the kiln has operated satisfactorily with little attention.

Fine carbon screened from the tailings is sent to Noranda's smelter.

It is intended to recycle solution from the tailings system when it eventually overflows.

Mine backfill is not produced.

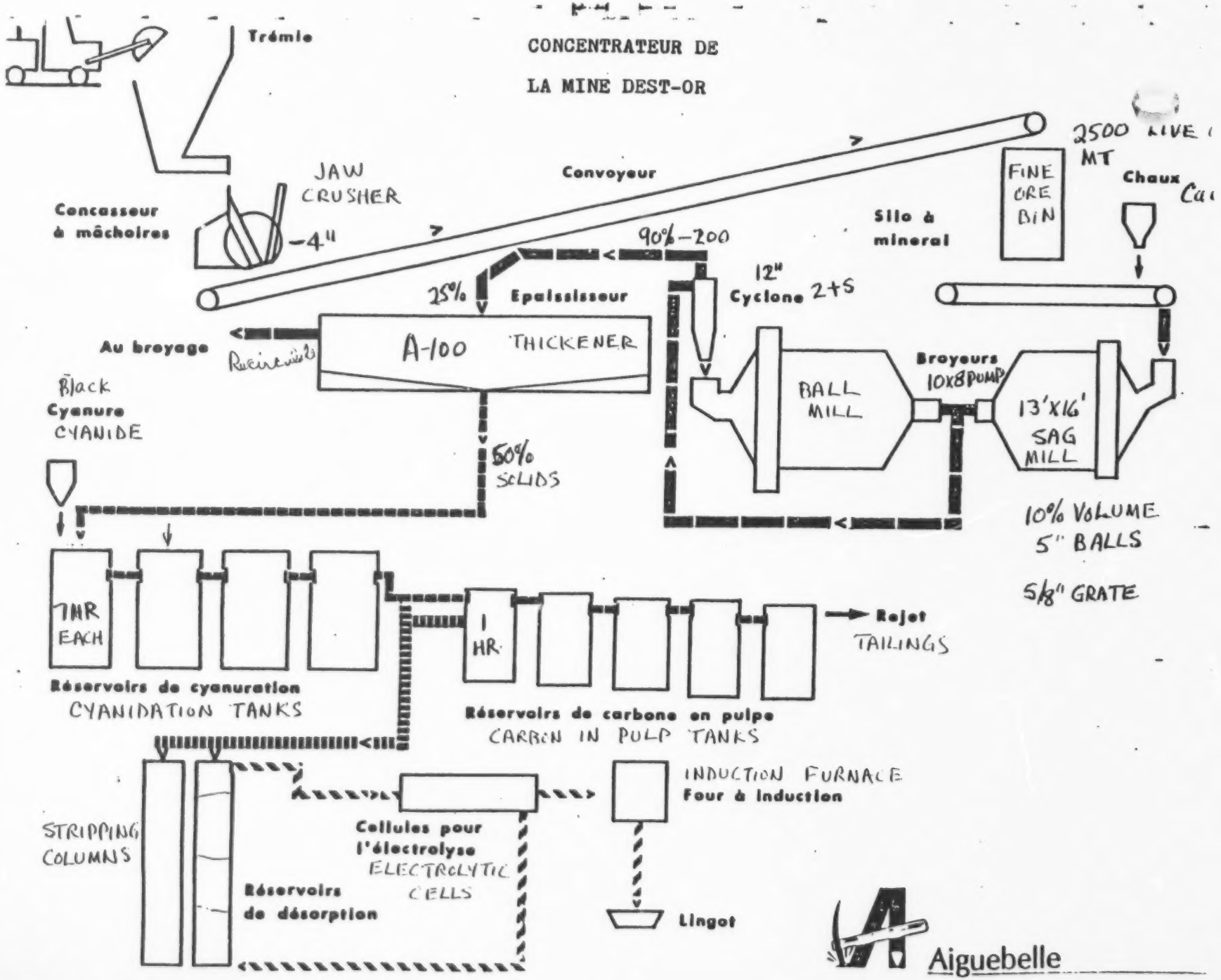
Only 17 men are required to operate and maintain the mill, including a mill superintendent, an operating-mechanical foreman, a metallurgist, three mechanics and two operators per shift (no shift boss) and three others.

The mill is compact and includes an office, laboratory and maintenance shop in the one building.

The mill may be expanded. It has been suggested that a substantial increase in throughput could be achieved by installing a cone crusher in the circuit to reduce the size of feed to the SAG mill, effectively turning it into a ball mill. (If this is true, it reflects on the efficiency of the SAG mill as now operated.)

Prince is quite pleased with performance of new mill, apart from some problems with the SAG mill liners.

CONCENTRATEUR DE LA MINE DEST-OR



Aiguebelle

NOTES OF MEETING WITH GEORGE HOPE REGARDING DOYON
MARCH 25, 1985

The writer met with George Hope at Kirkland Lake and also conversed by telephone with Jaques McMullen, General Superintendent of the Doyon and Est-Malartic mills.

Crushing

The original Kue Ken crusher has been replaced; it was found to be too light for the duty.

Ore from the mine is screened at 6 inch and the screen undersize is transported to the Est-Malartic Mill. The screen oversize is crushed and treated at Doyon and Noranda. The ore shipped to Est-Malartic is higher in sulphides and somewhat more refratory than the coarse fraction which is crushed and treated at Doyon and Noranda.

The ore is crushed to 5/8 inch.

Grinding and Leaching

Grinding is carried out in cyanide and lime solution.

Lime is slurried but not slaked and may not be as reactive as it should be.

An attempt is made to blend mill feed to maintain consistent proportions of pyrite and chalcocite in the feed, but it is not entirely effective. Lime levels are controlled by regular titrations, and vary from 1.5 to 2.5 lb./ton of solution, the higher value being associated with high sulphide levels. Overall, the pyrite content of the ore varies from 5 to 10%. (Drouin's paper indicates that cyanide levels are controlled at 1.0-1.5 lb./ton of solution. It also states that the grind has been reduced from 93%-200 mesh to 85%-200 mesh with no significant reduction in recovery. The recovery was said to fall off pretty markedly at coarser grinds).

Lime consumption has been reduced substantially since 1983 as a result of operating experience. When a "difficult" ore is encountered

lime additions are increased. Additions are then gradually decreased until recovery is affected.

Lead nitrate is added to the grinding circuit to complex sulphide irons and prevent them from forming thiosulphate and thiocyanates. Rates are as high as 0.5 lb./ton of ore. Lead salt addition is controlled by the Prussian Blue test. This test indicates the presence of ferric irons in solution. Lead nitrate addition is controlled to maintain a negative Prussian Blue test.

A major problem in the circuit is caused by chalcocite, which is quite soluble in cyanide.

Hope believes that cyanidation results cannot be predicted by laboratory testing due to the inability to simulate plant conditions, and, in particular, the effect of recycling barren solution to leaching.

Hope considers that the Eimco rake-lift agitators are not efficient air dispersers. He would recommend high efficiency mechanical agitators in a new plant.

Preaeration of the ore has not been tested in the plant; it might be helpful.

Doyon have outlined a new ore zone containing 400,000 tons at an average grade of 0.33 ounces per ton. According to McMullen, this ore is not totally different than that in the main zone but it is higher in sulphides and much of the gold is associated with the sulphides. Metallurgical testing of this ore has not been completed.

Comments

From Drouin's article and discussions with Hope and McMullen, it is apparent that the mill operators do not understand the complex chemical reactions in the mill circuit. Operators have learned to maximize gold extraction by generous additions of lime and lead nitrate.

A better understanding of the cyanidation chemistry would unlikely lead to higher gold extraction but could result in significant savings in lime and lead nitrate.

Although preaeration of the slurry could be very beneficial, it has apparently never been tested in operation.

According to George Hope, Lac is working with the Quebec government on an investigation into cyanide destruction using some method which is not new.

NOTES OF VISIT TO KIENA GOLD MINE

MARCH 27, 1985

R. Jolicoeur is Mill Superintendent.
Refer to attached flowsheets.

Ore Storage

Ore is crushed to 4 in. underground. On surface there are 2-650 tonne and one 1,700 tonne coarse ore bins. The steel bins are outdoors but are insulated. Freezing hasn't been a problem.

Grinding feed rate is measured with a Milltronics 4-idler weightometer which controls vibrating feeders. There is some difficulty in maintaining a steady feed rate because of the lag between the feeders and the weightometer and because the feeders are oversized (they were available from another operation).

Grinding

The 11.5 ft. x 18' semi-autogenous (SAG) primary mill came from Atikokan. It had been used as a rod mill and ball mill at Atikokan. Equipped with a 1,000 HP motor, it draws about 750 HP. The spare power is appreciated by the operators when the mill is started up. Ball charge, made up from 5 in. balls, is about 15% of mill volume. Capacity drops above or below this mill charge.

Water rate to the mill is automatically controlled in ratio to the mill feed.

Originally, the SAG mill was lined with chrome-moly lifters and shell liners. The discharge grate was rubber-covered steel. Because of difficulty with breaking bolts and the high noise level, 112 decibels, with the metal liners, Kiena replaced them with Skega rubber lifters and liners and the sound level has dropped to 90 decibels. The new liners were installed in December, 1984. No significant drop in mill capacity has been observed. Skega have guaranteed the liners for 8 months. Kiena believe that the liners will be good for 400,000 tonnes except for the last row at the discharge end which wear more quickly than the others.

Jolicoeur believes that the circuit would operate more efficiently if part of the primary cyclone underflow from the ball mill were directed to the SAG mill but he can't do this because the mill head sample is taken from the SAG mill discharge.

Operators have learned to control the SAG mill feed from the sound level (no instrument) and the power draw. A power recorder for the mill has been ordered.

The 10.5 ft. x 13 ft. ball mill is driven by an 800 HP motor which is overloaded (850 HP). It is also equipped with rubber liners. The mill operates in closed circuit with two-stage cyclone classifiers, grinding to 85% - 200 mesh. Media make-up is 2" balls.

Grinding is in cyanide solution and about 80% of the gold is dissolved in the circuit.

Thickening

Ground product is thickened to 50% solids in a 65 ft. dia. steel thickener prior to leaching. The unit is located outside and is not insulated. It sits on a 4 in. concrete slab on a gravel base. There is no tunnel. The underflow pipe is in a trench below the thickener. It is covered with plates and is accessible from above by draining the thickener. Underflow density is measured with a nuclear meter and the slurry is pumped with an SRL pump located inside the mill.

There has been no trouble with freezing in the thickener even when the plant was shut down for several days in December. However, Jolicoeur would prefer to have the thickener insulated. At Detour Lake the thickener is concrete and has been built up with mine waste for insulation.

Thickener feed is about 80°F and there is a temperature loss of approximately 10°F in the unit during the winter.

Leaching

Three leach agitators, said to be 30 ft. dia. x 33 ft. high (flowsheet indicates 33 ft. x 35 ft.), have a total retention time of 33 hr. Each is equipped with a Prochem agitator, with a single 10 ft. dia. rubber covered

propellor, driven by a 60 HP motor. The agitators are outdoors but tops are covered in.

The ore consumes quite a bit of oxygen. Operators found that they could introduce a maximum of 200 CFM below the impeller without causing bumping of the impeller. Two blades came off when a higher rate of air was used initially. To correct the problem, air lines have been introduced to the sides of the tanks near the bottoms and little or no air is introduced below the impellers. Approximately 400 CFM, vs. 150 CFM in some other mills, is added to each tank. Oxygen readings are taken three times weekly in the tanks. It has been found that 9 ppm. is adequate.

Black (calcium) cyanide is dissolved and pumped to the SAG mill and no. 1 leach tank to maintain a cyanide concentration (as NaCN) of 1 lb. per ton of solution in the thickener overflow. Slaked lime is added at the same points to maintain a level of 0.6 lb./ton of solution.

Cyanide consumption is 1 lb./ton (NaCN equiv.).

Lime consumption is 3 lb./ton.

No lead salts are used in leaching.

Gold Adsorption

There are two adsorption circuits.

Pregnant solution from the thickener overflow is pumped upflow through three static columns in series. Each column contains 3/4 tonne of carbon, which is advanced periodically counter-current to the flow of solution. Carbon loadings are in the 100-150 ounce per tonne range. Half of the gold produced is recovered in this circuit.

Gold in the leach discharge is adsorbed in five carbon-in-pulp tanks with a total slurry retention time of six hours. Carbon loadings are about the same as in the solution circuit.

The mill has been using Haycarb 6 x 16 mesh carbon but will be introducing Sutcliffe-Speakman 203 next. Carbon is pre-attritioned in a small agitator for 2 hours then screened on 16 mesh prior to use.

Stripping

Loaded carbon from both adsorption circuits is stripped in a stainless steel vessel that holds 2.7 tonnes of carbon. Stripping is carried out at

140°C and 60 psi (415 kPa). Three batches per week are stripped. Strip solution is 0.25% NaCN and 1% NaOH. Design called for 8 bed volumes of strip solution and 2 bed volumes of water wash during a 6 hour period. In practice it has been necessary to double the volume of strip solution and stripping time to achieve satisfactory results. The cause of the slow stripping has not been determined but it may be due to carbonate build-up on the carbon, which has occurred, or to the presence of iron in the strip solution (Dest-Or was reported to have the same problem). Kiena is considering acid washing the carbon before, rather than after stripping in an attempt to improve the stripping.

Carbon Washing and Regeneration

Following stripping all carbon is washed with 5% nitric acid in a fiberglass vessel.

Originally all carbon was regenerated in a combination dryer and regenerator operated at 280°F and manufactured by Custom Equipment Co. Due to the failure of temperature controls on the unit, part of the refractory melted down and the unit hasn't been in operation for 1 1/2 months.

Total carbon in the whole mill circuit is 12 tons.

Electrolysis

Gold is electrowon from strip solution in an electrolytic cell that holds 20 steel wool cathodes.

Smelting

Cathodes are smelted twice weekly in an electric induction furnace to produce bullion. Smelting takes about 4 hours using two men. Slag is presently being stockpiled for possible sale. It is known that gold in the slag can't be recovered by returning it to the grinding circuit.

Filtration

About 80% of the leach slurry from the carbon-in-pulp tanks is filtered on disc filters to recover free cyanide and to reduce the amount of cyanide going to the backfill plant (filter capacity is inadequate to

handle the total flow). The filters came from Elliott Lake and are stainless steel. It was intended to wash the filters periodically with acid in the filter boot but this has not been necessary. Occasionally individual filter sections are taken out and washed with acid.

Cyanide Destruction

Filter cake is repulped and together with unfiltered leach slurry, is treated in one tank with sulphur dioxide, air and copper sulphate to destroy free cyanide. This is necessary to meet regulations for cyanide levels in mine backfill and to render the pyrite floatable in subsequent processing. Mine regulations are apparently being met.

The cyanide destruction circuit has been operating for a short time and details of the process were not discussed.

Backfill Plant

Sand for mine backfill is recovered from the waste solids by 2-stage cyclone classification using overflow densities of 15% and 30% solids in the primary and secondary stages, respectively. The circuit is said to be capable of producing a sand yield of 55-60%, with a percolation rate of 5-6 in./hr. This is an unusually high yield for a find grind and is probably attainable because none of the primary cyclone overflow is recycled and the secondary cyclones are smaller in diameter (6 in.) than those used in many backfill plants.

Primary cyclone overflow is thickened to 45% solids and pumped to the tailings pond. Occasional sliming in the thickener is controlled with a small quantity of Percol 351 flocculant.

Pyrite in the sand may be recovered by flotation in a 100 cu. ft. cell and be returned to the grinding circuit. This cell was not operating at the time of the visit.

Tailings Disposal

Waste solids are pumped about 3 km. to a large tailings disposal area on the mainland. The area is relatively flat, requiring a dam of mine waste and other material on all sides. Effluent from the pond will

go to a large polishing pond which has been sealed with clay from the site and polyethylene.

The tailings line and ponds cost $5 million but provide storage for several years of mill operation.

No overflow is expected for 1 1/2 to 2 years.

Assaying

All mill solutions are assayed in the mill using atomic adsorption. Mine samples and mill solids samples are presently being sent to a custom assay office; however Kiena intend to set up their own fire assaying facilities in the near future.

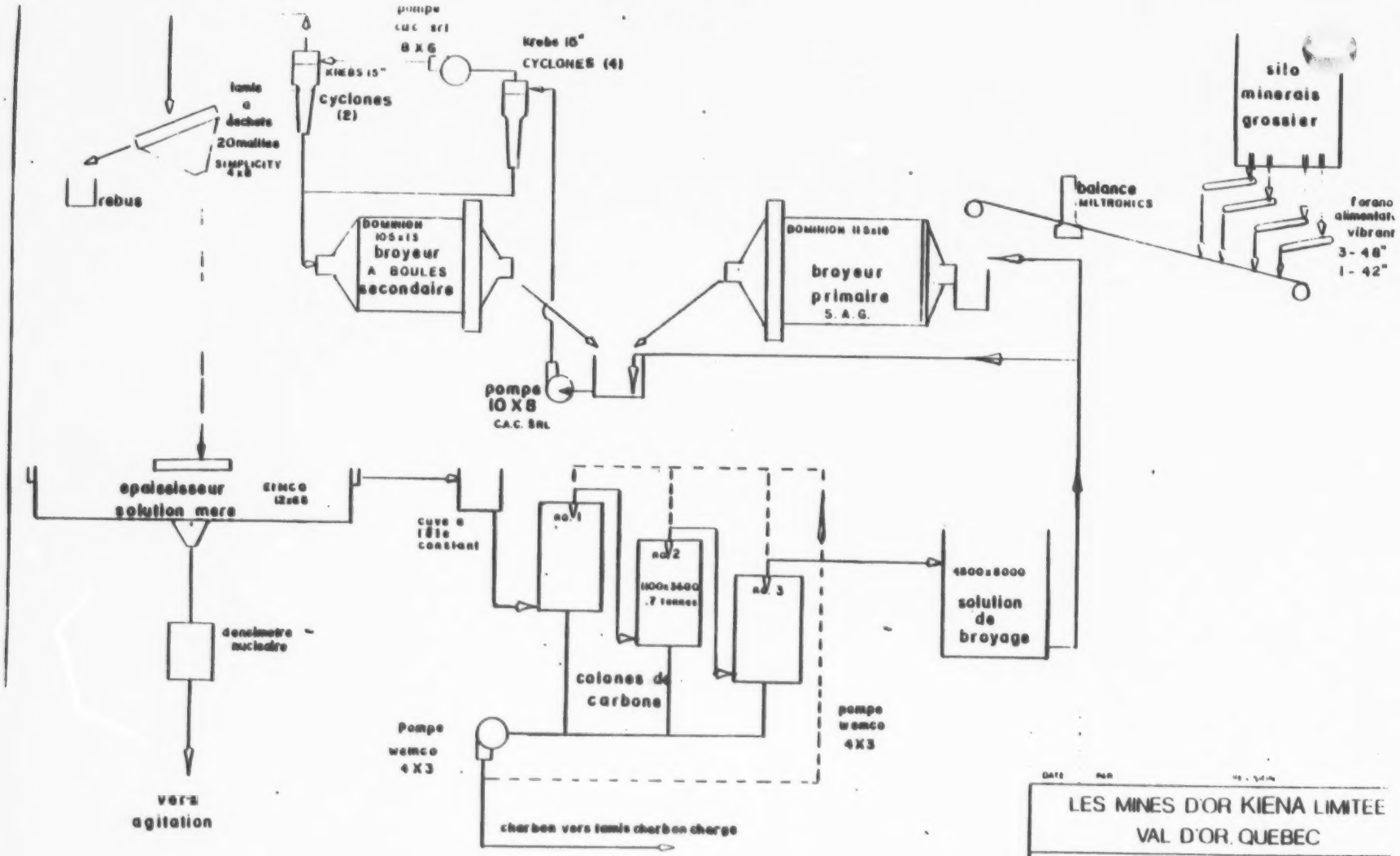
Mill Staff

Total personnel in the mill for operations and maintenance is only 23, distributed as follows:

Mill superintendent	1
Metallurgist	1
General foreman	1
Operators	12
Technicians - refiners	3
Mechanics	4
Electrician	1
Total	23

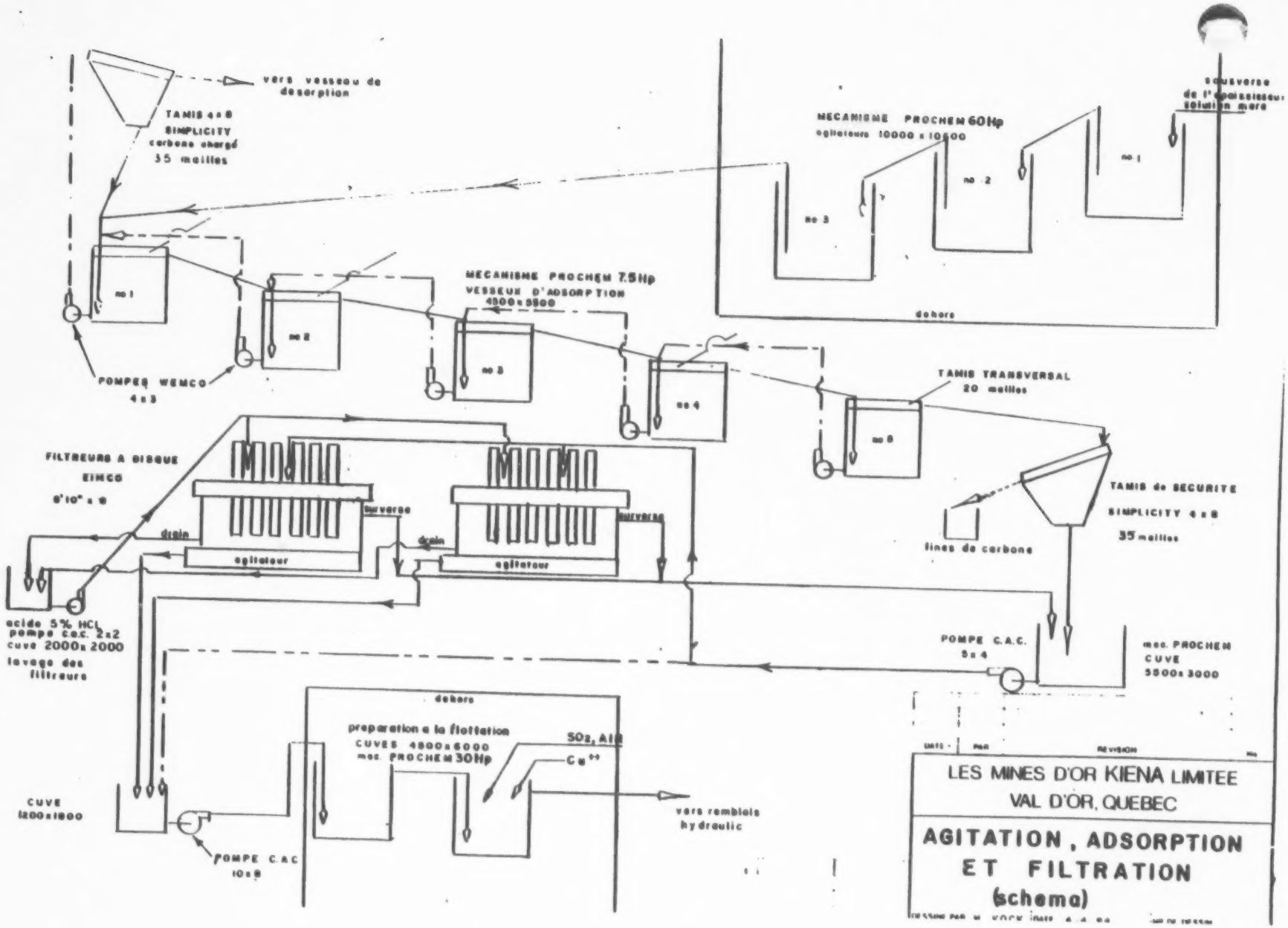
Assaying is not included.

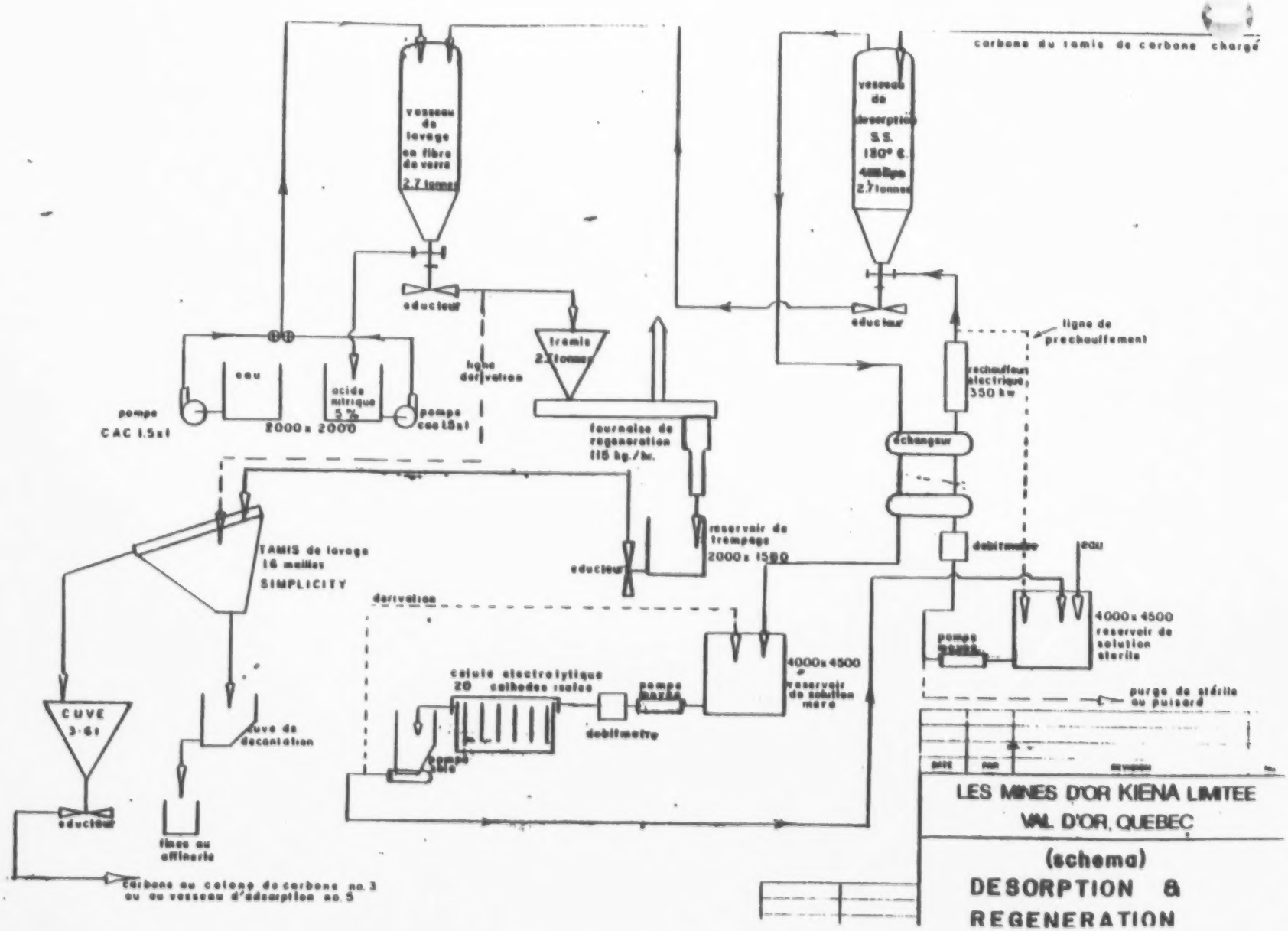
Almost all maintenance is carried out in a shop located in the mill.

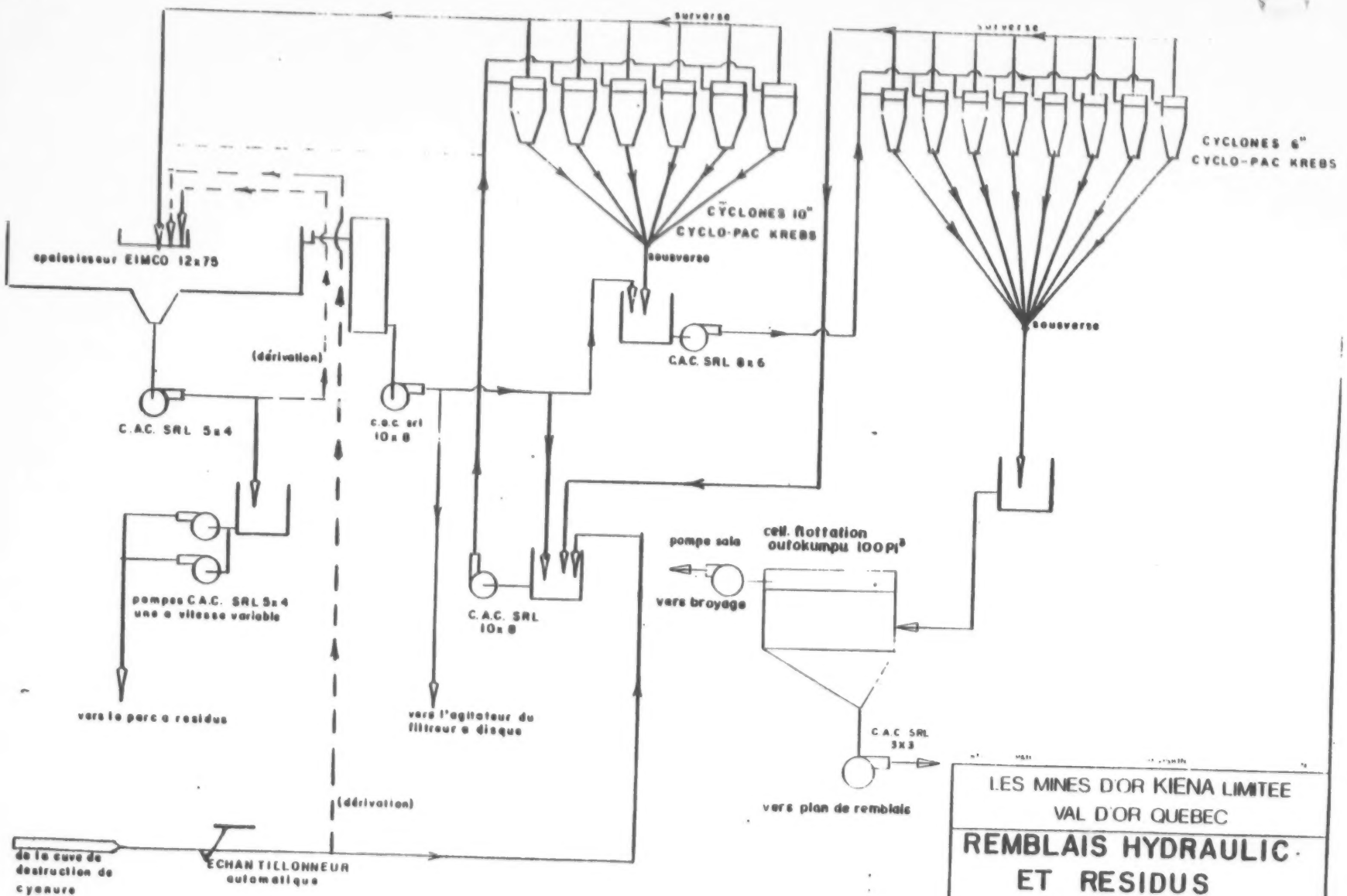


LES MINES D'OR KIENA LIMITEE
VAL D'OR, QUEBEC

SCHEMA DE BROYAGE


vers vesseau de desorption
TAMIS 4 x 8
SIMPLICITY
carbone chargé
35 mailles
MECANISME PROCHEM 60Hp
agitateurs 10000 x 10500
no 1
no 2
no 3
déverse de l'épaississeur solution mere
dehors
MECANISME PROCHEM 7.5Hp
VESSEUX D'ADSORPTION
4500 x 5500
no 1
no 2
no 3
no 4
no 5
POMPES WEMCO
4 x 3
TAMIS TRANSVERSAL
20 mailles
FILTREURS A DISQUE
EIMCO
8'10" x 8
drain
déverse
agitateur
TAMIS de SECURITE
SIMPLICITY 4 x 8
35 mailles
fines de carbone
acide 5% HCL
pompe cac. 2x2
cuve 2000x 2000
lavage des filtreurs
POMPE C.A.C.
5 x 4
mec. PROCHEM
CUVE
5500x 3000
dehors
preparation a la flottation
CUVES 4800x6000
mec. PROCHEM 30Hp
SO2, Air
Cu++
vers remblais hydraulic
CUVE
1200x1800
POMPE C.A.C
10 x 8
LES MINES D'OR KIENA LIMITEE
VAL D'OR, QUEBEC
AGITATION, ADSORPTION
ET FILTRATION
(schema)


carbone du tamis de carbone chargé
vesseau de lavage en fibre de verre 2.7 tonnes
vesseau de desorption S.S. 130° C. 2.7 tonnes
ligne de prechauffement
eau
acide nitrique 5 %
pompe CAC 1.5x1
2000 x 2000
pompe eau 1.5x1
ligne derivation
tamis 2.7 tonnes
fournaise de regeneration 115 kg./hr.
rechauffeur electrique 350 kw
echangeur
TAMIS de lavage 16 mailles SIMPLICITY
reservoir de trempage 2000 x 1500
debitmetre
4000 x 4500 reservoir de solution sterile
derivation
cellule electrolytique 20 cathodes isolés
4000 x 4500 reservoir de solution mère
CUVE 3-6t
cuve de decantation
fines ou affinerie
purge de stérile au puisard
LES MINES D'OR KIENA LIMITEE
VAL D'OR, QUEBEC
(schema)
DESORPTION & REGENERATION



LES MINES D'OR KIENA LIMITEE
VAL D'OR QUEBEC

REMBLAIS HYDRAULIC ET RESIDUS (schema)

SIGMA USES EFFICIENT MINING METHODS TO RECOVER LOW-GRADE ORE

High gold prices, efficient mining methods, and the gradual depletion of higher-grade ore have motivated Sigma Mines (Quebec) Ltd. to initiate an expanded exploration program, with emphasis on the lower-grade ore surrounding the existing mine. The program has revealed that, at prevailing gold prices, the mine can operate for five years at the current annual mining rate of about 480,000 st of ore grading 0.115 oz/st, followed by 10 years of a gradual decrease in production and ore grade. The mine, located about 1 mi east of Val d'Or, has operated continuously since 1937 and occupies an area that covers about 1,470 acres.

Stope mining accounts for the bulk of the extraction at Sigma, producing about 91% of the ore, while stope preparation and development work produce 6% and 3%, respectively (these figures vary from year to year). Mine development is currently distributed between the 8th and 39th levels. Major programs in the mine plan include the development of the "0-16" zone on the lower levels and "S" zone on the 29th level.

The Sigma milling facilities, located on the mine site, process about 1,400 st/d of ore with a recovery rate of 96.4%. Total costs per short ton of ore milled is about $C35.00.

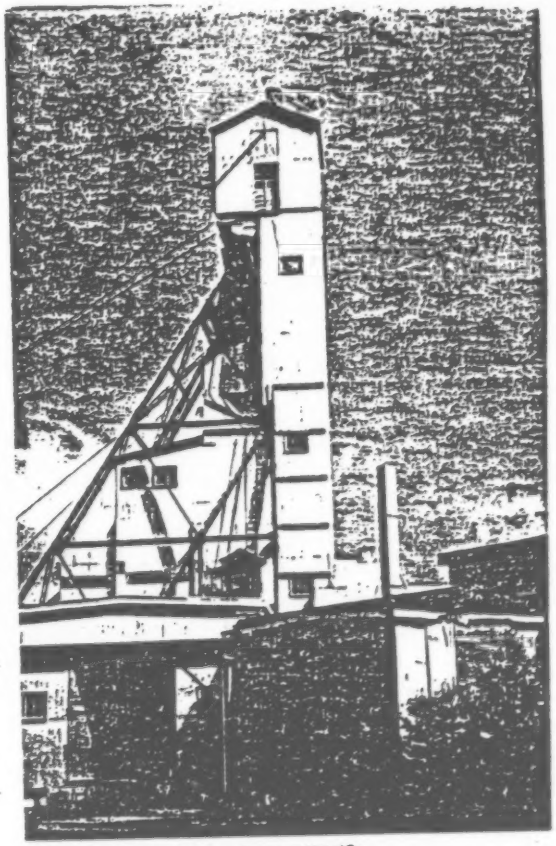
The mine is located within the Archean Malartic Group's steep-dipping, east-striking, ultramafic flows, basalts, tuffs, agglomerates, and andesite rocks. These rocks were slightly overturned and intruded by the Bourlamaque batholith to the north and Lamaque plug to the south.

Gold mineralization occurs in quartz-tourmaline-pyrite shear-, flat-, and stringer-type veins that fill fractures and shear zones centered on the C-type diorite dyke complex. Gold is mostly invisible and is found to occupy microfractures within the pyrite crystals and quartz grain boundaries, although visible nuggets are often found in the veins.

Vein material of the south-dipping network of shear zones was introduced intermittently while the shears were active, as indicated by the separation and crosscutting relationships of some veins. The sheared veins, as they are called, vary in thickness from a few inches to 5-6 ft and occur as lenses having a lateral and vertical extension up to several hundred feet.

Flat veins dip to the west about 20° and traverse the undisturbed host rock between shear zones. The veins are vertically displaced by weak shears and often terminate against the sheared veins. Gold content is usually higher where pyrite is most abundant.

Dyke stringers lie almost perpendicular to the walls of the G-porphyry dykes and form an *en echelon* pattern that dips to the west at about 40°. The stringers are identical in composition to the sheared and flat veins, and gold content seems to be a function of wall-rock alteration and pyrite content.



View of the Sigma mine's headframe at Val d'Or.

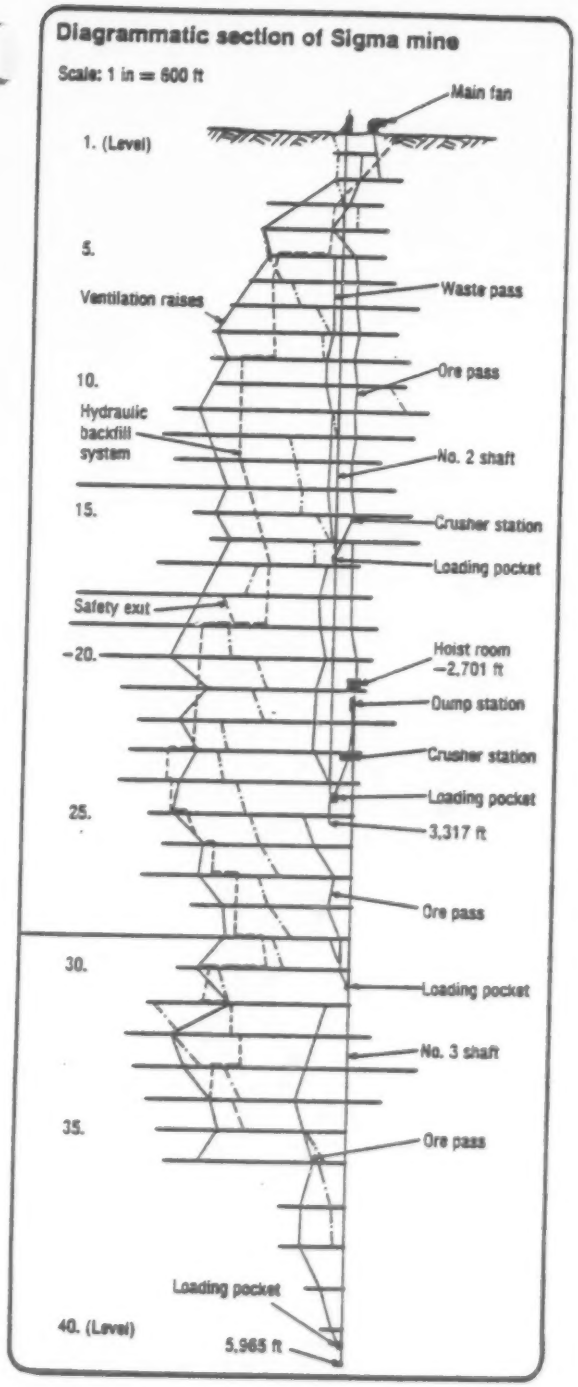
THE SIGMA MINING PROGRAM

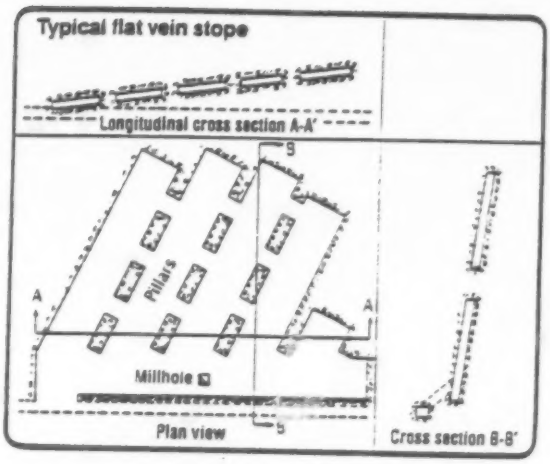
The Sigma mine handles annually about 480,000 st of ore on a two-shift-per-day, five-day-per-week work schedule. Mining is carried out using three variations of stoping to suit the attitude and limits of the ore zone. These include shrinkage stoping for veins that average about 8 ft in thickness and dip about 70°, and feldspar porphyry dykes averaging 20 ft in thickness and dipping about 80°; cut-and-fill stoping for 8- to 12-ft wide, 70°-dipping, quartz-tourmaline veins contained within relatively weak zones in deeper horizons; and flat vein stoping for relatively horizontal veins that average 6 to 24 ft in thickness.

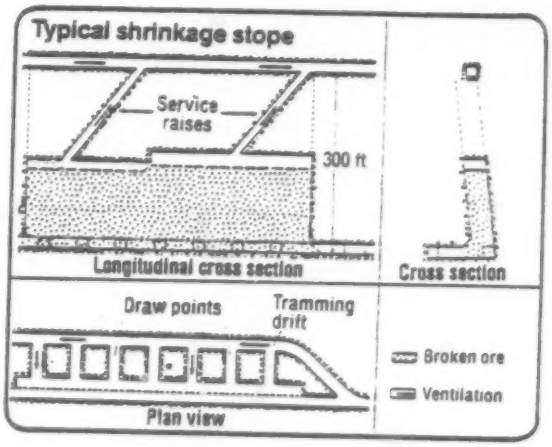
For mine development, 8 x 8-ft drifts and crosscuts are advanced at rates of 8-10 ft/shift using C.I.R. JR 300 drills mounted on single-boom Long Tom drill jumbos. An Eimco 12B loader is used for mucking, while 1½-st Mancha's Little Trammer Type "B" locomotives and 2-st side-dump cars are used for hauling. Virtually all blasting is done with Anfo.

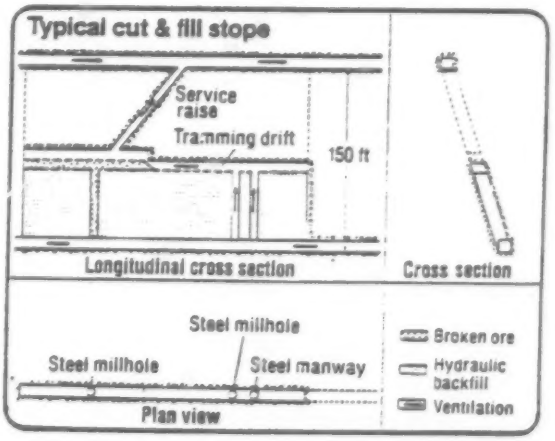
Additional equipment at Sigma includes Gardner-Denver FL 63 and Sico 240 jacklegs; 36 x 42-in. Esco, Pacific, and Joy and Sorel box- and hoe-type scrapers; Pinkrose Rotorair slushers; and four 35-ft³ Cavo 310 autoloaders. For hauling, Sigma uses 30 1½-st Mancha's Little Trammer Type "B" locomotives and one 3½-st Titan BNX locomotive, 205 44-ft³ side-dump cars located at the No. 2 shaft levels, and 131 36-ft³ side-dump and 76 32-ft³ end-dump cars located at the No. 3 shaft levels.

Ventilation in the Sigma mine is provided by a 60-in.-dia Joy Oxivane axial flow fan rated at 150,000 ft³/min of air.









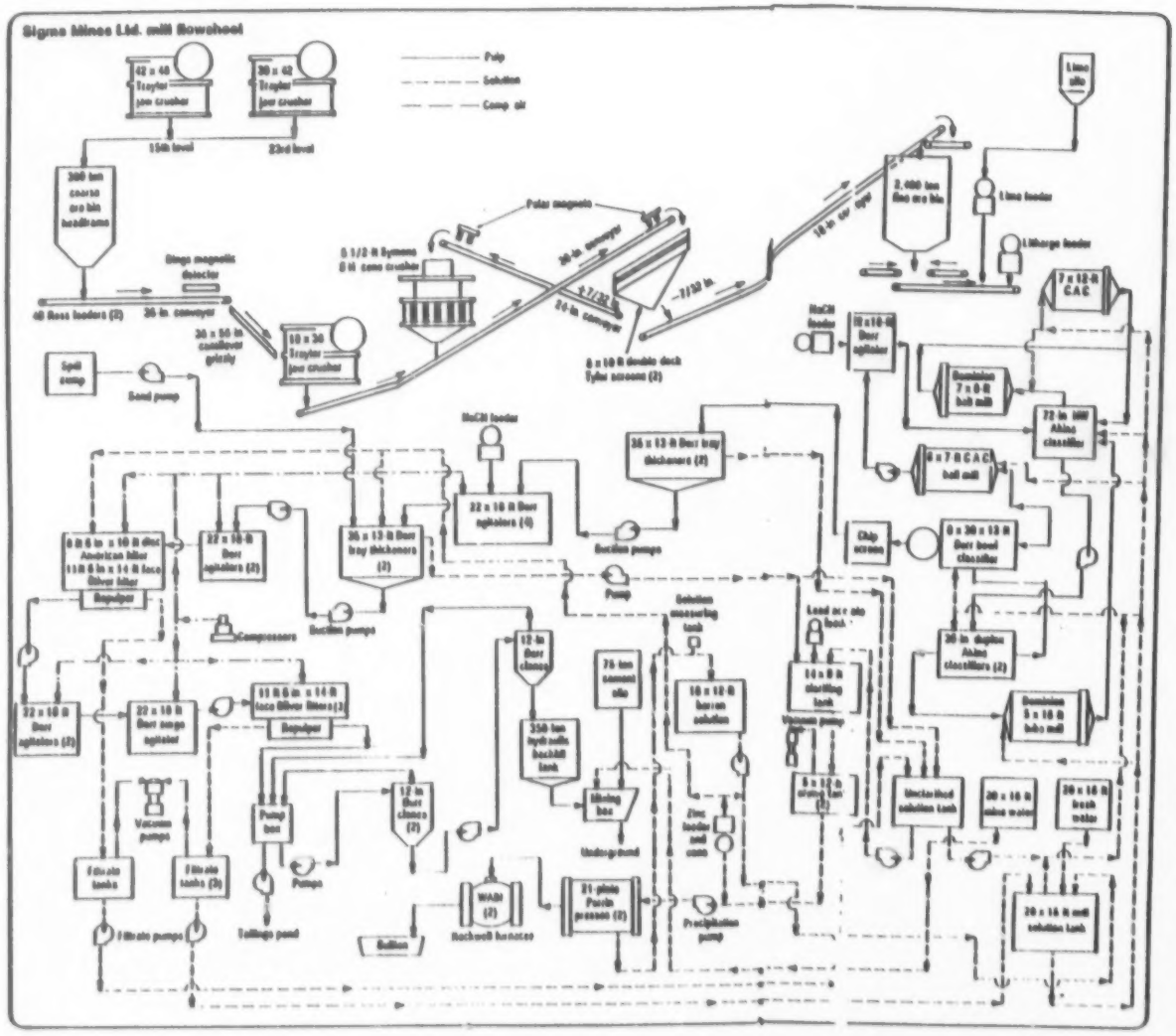
The fan is located on the surface and is driven by a 200-hp motor. A second Sheldons 60-in.-dia flow fan, rated at 80,000 ft³/min of air, is on standby.

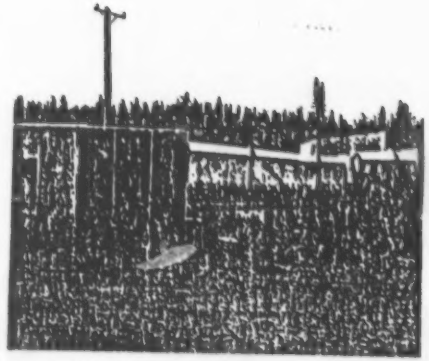
The ventilation system consists of ventilation raises that connect to each mine level. Regulators located at each level control the volume of air released. The air circulates through drifts and crosscuts and is upcast through the main shaft, ore and waste passes, and fill raises. Auxiliary ventilation, consisting of 12-in.-dia and 16-in.-dia axial flow fans, is used in working areas not affected by the main system.

ORE PROCESSING

A 42 x 48-in. and a 30 x 42-in. Traylor jaw crusher, located below the 15th and 23rd levels, respectively, reduce ore to minus 6-in. The crushed ore is hoisted to the surface

Sigma Mines Ltd. mill flowsheet


Pulp
Solution
Comp. air
42 x 60 Traylor jaw crusher
30 x 42 Traylor jaw crusher
15th level
23rd level
300 ton coarse ore bin headframe
Dings magnetic detector
48 Ross loaders (2)
36 in. conveyor
36 x 50 in. cantilever grizzly
10 x 36 Traylor jaw crusher
5 1/2 ft Symons 0 ft cone crusher
Pulsi magnetic
36 in. conveyor
30 in. conveyor
10 in. ore pipe
2,400 ton fine ore bin
Lime silo
Lime feeder
Lime feeder
7 x 12-ft CAC
Sand camp
Sand pump
6 x 10 ft double deck Tyler screens (2)
10 x 10-ft Dorr agitator
Dominion 7 x 8 ft ball mill
72 in. HW Akins classifier
8 x 7 ft CAC ball mill
Chip screen
8 x 30 x 13 ft Dorr bowl classifier
36 x 13 ft Dorr tray thickeners (2)
22 x 10 ft Dorr agitators (4)
Suction pumps
Pump
Solution measuring tank
Lead ore silo feed
36 in. duplex Akins classifiers (2)
Dominion 5 x 10 ft rod mill
8 ft 6 in. x 10 ft disc American filter
11 ft 6 in. x 14 ft face filter filters
Repulper
27 x 10 ft Dorr agitators (2)
Compressors
Suction pumps
12 in. Dorr classifier
75 ton cement silo
10 x 12 ft barren solution
14 x 8 ft clarifying tank
Vacuum pump
22 x 10 ft Dorr agitators (2)
22 x 10 ft Dorr surge agitator
11 ft 6 in. x 14 ft face filter filters (2)
Repulper
350 ton hydraulic backfill tank
Mixing box
Underground
5 x 12 ft drying bin (2)
Unclassified solution tank
20 x 16 ft mine water
20 x 16 ft fresh water
Vacuum pumps
Pump box
12 in. Dorr classifier (2)
Pumps
Filtrate tanks
Filtrate tanks (2)
Filtrate pumps
Tailings pond
Boilers
WAH (2)
Rockwell furnaces
21 plate Portis pressure (2)
Zinc feeder and mix
Precipitation pump
20 x 16 ft mill solution tank



A 60-in. slip mixed flow fan supplies ventilation at a rate of 150 000 ft³/min at 40 in. to the underground workings

using 5-st skips and stored in a 300-st coarse ore bin.

The surface crushing plant consists of an 8 x 36-in. Traylor jaw crusher with a 2½-in. setting operating in open circuit, and a 5½-ft Symons Short Head crusher set at ¾ in. and operating in closed circuit with two 5 x 10-ft, double-deck Ty-rock screens. Discharge material is conveyed to a 2,400-st-capacity fine ore bin.

Grinding is done by a 7 x 12-ft Canadian Allis Chalmers (C.A.C.) overflow-type ball mill operating in open circuit with a media charge of 100% 3½-in. forged steel balls. Ball-mill discharge goes to a 72-in. Akins spiral classifier operating in closed circuit with a 7 x 8-ft Dominion ball mill. Secondary classification and grinding is done with two 36-in. Akins duplex spiral classifiers and a Dominion 5 x 10-ft Jobe mill using 1½-in. balls as grinding media. Tertiary classification and grinding is carried out by a Dorr bowl classifier and a 6 x 7-ft C.A.C. ball mill. Mill discharge is sent to an agitator for a longer retention time, with its overflow being returned to the primary classifier. Final product is 60% minus 200 mesh.

Ore treatment consists of a straight cyanidation circuit with primary thickening and agitation with pulp at 60% solids, secondary thickening and agitation with pulp at 61% solids, primary filtration, tertiary agitation, and final filtration repulped with barren solution and washed. Total contact time is about 37 hr. Tailings are discharged at 62% solids through a 6-in., wood-stave pipe to a disposal area on the property.

Sodium cyanide is added and maintained at 0.35 lb/ton of ore to the primary thickener and the special agitators. Quicklime is also added to the primary ball mill feed at a rate of 1.34 lb/ton of ore.

For precipitation, the Merrill Crowe process is used with simultaneous clarification and de-aeration. Two 36-in. Perrin presses are employed alternately during 14 day precipitation cycles. Precipitate from the presses is mixed with flux (28% borax, 17% silica, 40% nitre, and 15% flourspar) and refined in two Wahl rotating bullion furnaces equipped with oil and air mixture burners.

M.P.S.

Notes of Visit to Campbell Red Lake Mine, March 28, 1985

Met with J. Scott Roberts, Mill Superintendent.

Dennis Molloy is Assistant Mill Superintendent.

Ross Melville, formerly with Equity Silver, is Metallurgist.

Crushing and Grinding

Primary crushing to 8 inch is underground. Three-stage crushing in the mill reduces the product to 1/4 in. The "B" grinding circuit was added recently. The optimum grind is in the range 73-80% minus 200 mesh. Finer grinding results in lower recovery in flotation and on the blanket. The grind averaged 73.4%-200 mesh in 1984.

Primary ball mills have Ni-Hard liners and the secondary mills rubber liners.

The narrow 16 in. x 24 in. Denver jigs are said to operate better than the 24 in. x 36 in. jigs that were previously used.

Flotation

Two banks of Denver DR flotation cells were installed three years ago. In each bank concentrate from the first 6 cells normally goes to the blanket and then to the roaster, from the next 6 cells directly to the roaster and from the final 4 cells middlings are returned to the grinding circuit. Double launders in some of the cells provide flexibility of operation.

Copper sulphate is added to conditioners ahead of the second stage of flotation. Excess copper sulphate addition depresses stibnite. Antimony will cause sintering in the roaster bed if present in concentrations well above 1%. In the past two years antimony in the concentrate has averaged only 0.11%.

Only 60% of the sulphur in the ore is recovered by flotation. Sulphur content of the ore at lower mining levels has been dropping and Campbell is having difficulty maintaining the sulphur in the concentrate high enough (17-18%) to obtain optimum temperatures with an autogenous roast. It may be necessary to add elemental sulphur or an alternative fuel to the roasters in the future.

Flotation reagents, except copper sulphate, are added in the grinding circuit and the cyclone overflow.

Thickener

Flotation tailings are thickened to 65% solids in a new 90-foot thickener that went into operation in late 1984. It is located outside. In the first winter there was no difficulty in operating it but some flocculant (Percol 351) was necessary in the winter to control sliming.

Cyanidation of Flotation Tailings

Flotation tailings are prearated in two tanks (15 hr.) and cyanided in 11 Dorr agitators (45 hr.). Solutions will foul if the pulp is not preaerated. No lead salt is used in preaeration.

Cyanide is controlled at 1.5 lb/ton of solution by titrations.

Lime is added to maintain a pH of 10.0-10.5 in the fourth agitator.

Roasting

The roasting plant has been well described in published literature.

The roaster operates about 80% of the time. It runs for 6-7 days and then is shut down for two days. Capacity is 100 TPD of concentrate.

The roasters are now operated on computer control from an air conditioned room, with one operator per shift.

Filter bags are Draylon T. These are replaced every two years whether they need it or not.

Calcine Treatment

Quenched calcine contains quite a bit of free gold. It is recovered on a new 25-foot endless rubber belt and a James (similar to Wilfley)table. Table tailings are dewatered by thickening and disc filtration. This removes most of the salts in the calcine.

The calcine is reground to 70%-325 mesh, combined with ESP dust and cyanided in a conventional circuit.

Tailings assay about 0.25 oz/ton. Tests have shown that finer grinding would be helpful. Gold is higher in the finer fractions than the coarse, and is probably in unreacted arsenopyrite. Free gold has been seen in the residue; it must be tarnished.

Campbell are considering carbon-in-pulp treatment for the calcine residue. A 3-foot Reichert spiral is being tested on the final calcine residue to see if some gold can be scavenged from it. Spiral capacity is 5 tons/hr.

Gravity Concentrates

Three times weekly the various gravity concentrates are batch-tested by two men in one shift using two shaking tables. The first table separates gold and sulphides from the gangue and the second separates gold from sulphides. Middlings are returned twice.

The final gravity concentrate, containing 75-80% gold is smelted, producing 915 fine bullion.

Precipitation

Pregnant solution from both leaching circuits is clarified and gold recovered by the Merrill-Crowe process. Almost all barren is discarded.

Backfill

About 30% of the waste solids are recovered for mine backfill, insufficient to provide all the sand required in the mine.

Arsenic

The crude arsenic produced is shipped to Mineral Research and Development, Harrisburg, N.C.

Tailings

Fine solids are pumped to a new waste disposal area. It consists of a waste solids area with a clear water pond holding 185 days' solution, followed by a polishing pond. There is no recycle to the mill from this area. Effluent from the polishing pond runs into Balmer Lake year round, along with Dickenson's effluent. Balmer Lake is considered to be part of the two mines'

Waste management systems. The mines have 1½ years to demonstrate that the effluent from the system can meet government standards. If not, effluent treatment will be necessary. Campbell may use sulphur dioxide from their roaster to treat effluent if it is necessary.

A scavenger circuit to recover gold from the polishing pond effluent with carbon is in operation. Two parallel lines of four adsorption columns each handle the total effluent from the pond. An atmospheric stripping circuit and electrolytic cell went into operation recently. This circuit is still in the experimental stage and it is too early to tell whether it will be very profitable.

The carbon is picking up copper as well as gold.

Reagent Consumptions - 1984

Reagent	Lb/Ton
Primary grinding - steel balls	0.93
Secondary grinding steel balls	1.17
Cyanide - NaCN	2.50
Lime - CaO	1.40
Xanthate	0.33
Frother	0.051
Zinc	0.072
Hydrochloric acid	0.077
Filter Cel	0.086
Percol 351	0.014

Instrumentation

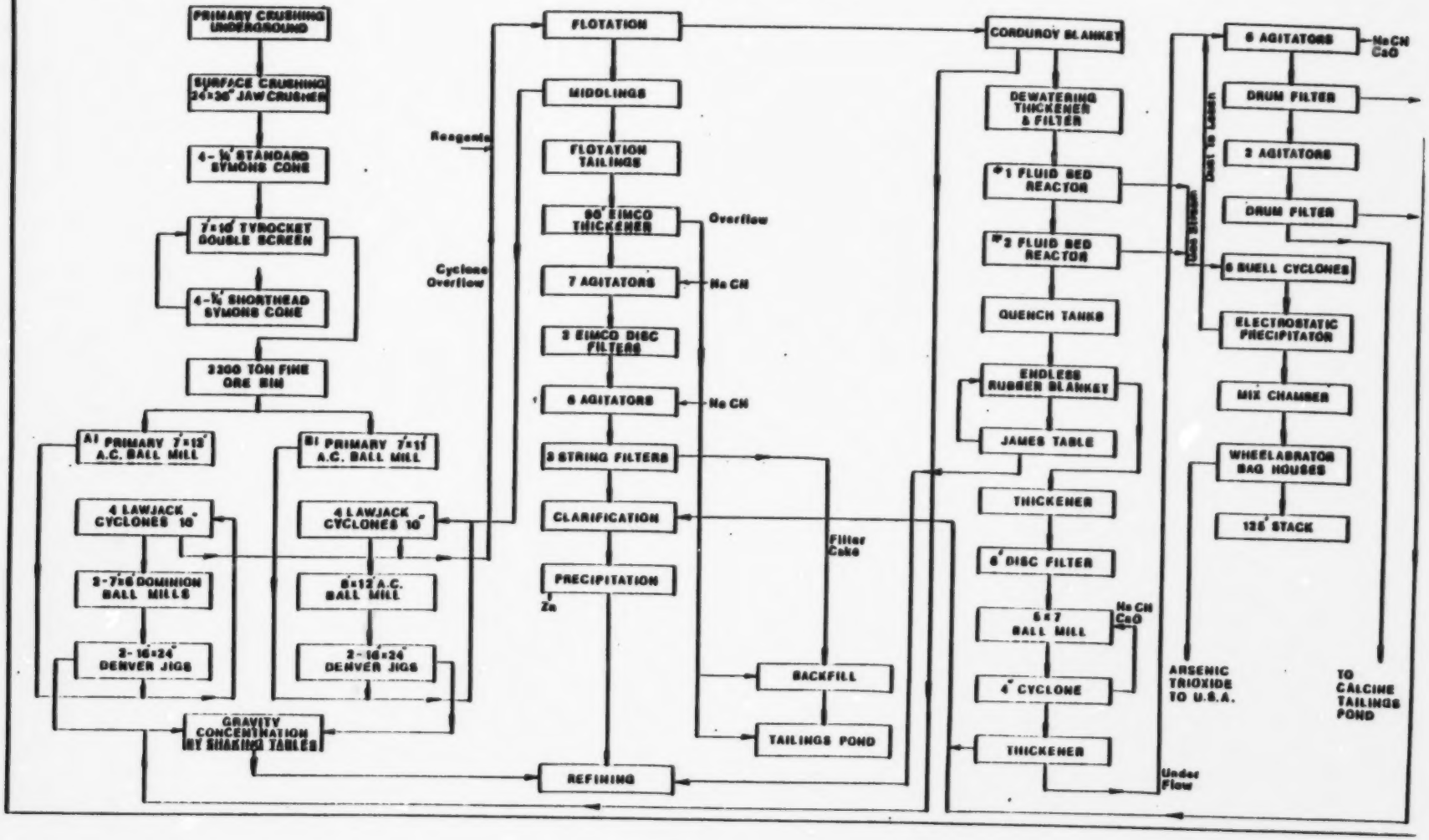
Apart from the roasting plant there is little instrumentation in the plant.

A study is underway on the feasibility of automating the whole plant. It will probably show that savings will mainly be in labour rather than in reagents or higher recovery.

Campbell tested an in-line toroidal conductivity sensor in the mill but found that, at high pH the correlation between conductivity and pH was not satisfactory.

CAMPBELL RED LAKE MINES LIMITED

MILL FLOW SHEET



NOTES OF VISIT TO DICKENSON MINE
FEBRUARY 21, 1985

Met with Larry Connell, Mill Superintendent.
Refer to attached flowsheet.

Mill Capacity

Now at 800 TPD, the capacity could be raised to 1000 TPD with little expense. The fine ore bin, with only 900 tons capacity would have to be increased or crushing carried out every day. There appears to be plenty of ore available at lower levels and the mill capacity may be raised to as high as 1500 TPD.

Crushing and Grinding

There are pockets of pyrite in the ore. Pyrite and arsenopyrite contents of the ore vary considerably. Gold grade from various zones ranges from 0.1 to 1 ounce per ton.

The ore is crushed to 3/4 in.

Grinding mills are not new and there is no instrumentation in the circuit.

The ore is ground to 70% - 200 mesh. In lab tests finer grinding, even to 600 mesh, did not result in higher gold extraction.

Cyanidation

Leaching begins in the grinding circuit. Cyanide (calcium cyanide) solution is fed to the rod mill and three other points to maintain cyanide levels (sodium cyanide equivalent) of 1.5 and 1.0 lb. per ton of solution at the head end of the circuit and at precipitation respectively. Lime is added to maintain pH of 10.5 to 11.0.

Lead nitrate is also added to the rod mill and at two points in the leach circuit to control the reducing power below 4.5 (450?). In the absence of lead salt the sulphur compounds in solution compete for the available oxygen. On some days no lead salt is required. Connell believes that if enough air could be introduced into the pulp the lead might not be necessary.

Only one air-lift agitator remains in the circuit. The others have been replaced with Prochem agitators. These have a fluid drive. One new agitator has been in service for 2 years. It is now starting to flex. Whether the blades are worn or damaged isn't known because the tank hasn't been drained since the agitator went in. Each leach tank has 4 side air jets to improve aeration.

Extraction has improved with the new agitators and improved aeration.

To increase the plant capacity double-stage filtration has been discontinued and all filters are used for single-stage filtration. Formerly, with two-stage cyanidation, there was a "bump" in extraction when the first stage filter cake was repulped in fresh solution, suggesting that solution fouling was hindering gold extraction.

With only one filtering stage, soluble losses are now higher, around 1.3%. To reduce soluble loss to about 0.3% Dickenson is now installing a carbon-in-pulp (CIP) circuit to scavenge gold from the slurry between filtration and flotation. The circuit will consist of two stage extraction (4 hours), atmospheric stripping with provision for nitric acid washing and electrolytic recovery of gold. Carbon regeneration may be added later.

Eventually the old filters will be replaced by a multi-stage CIP circuit. Presumably, the Merrill Crowe system will be replaced by static carbon columns.

All barren solution is bled to tailings.

Tailings

Tailings are pumped to a large pond with a decant structure. The decant flows to a polishing pond from which the water percolates out through a rock dam.

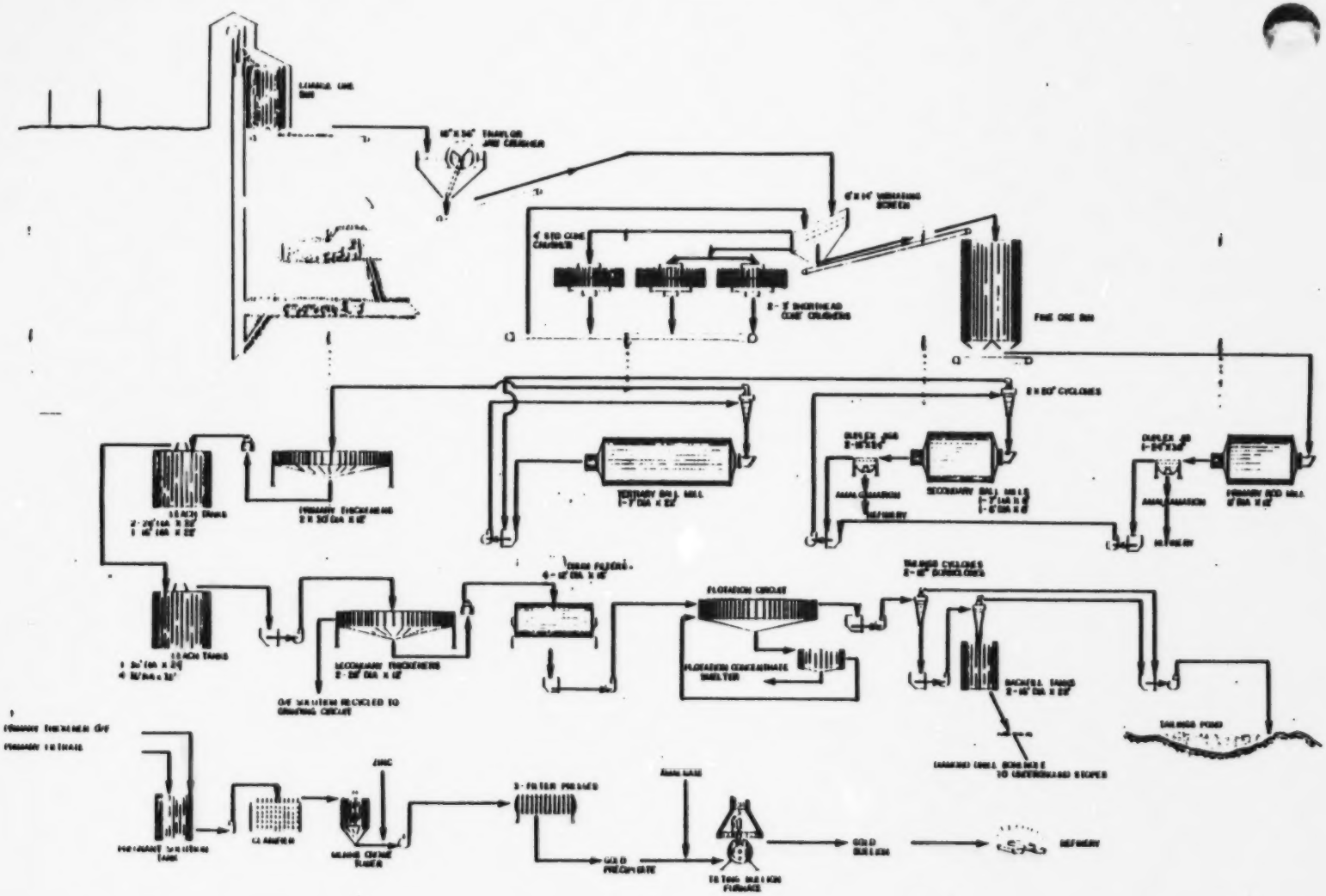
Effluents from Campbell and Dickenson flow into Balmer Lake where natural degredation of cyanide occurs. Balmer Lake is considered to be part of the two mines' tailing system. Outflow from Balmer Lake, which flows year-round meets the 2 ppm guideline set by the Province. In fact. in 1984 it averaged less than 1 ppm. If the mines can demonstrate that they can meet regulations year-round they will not have to install a chemical system for cyanide destruction.

Water from Balmer Lake is recycled for process water.

Dickenson tested carbon adsorption on the polishing pond effluent but because of dilution in the systems loadings were very low.

Backfill

The Joy Tailspinner has been tested on mine backfill. Because it produces a slurry with higher density than by cycloning, the Tailspinner product doesn't segregate and fine cement isn't lost, as happens at times with the cyclone product. Unfortunately, the Tailspinner product is too thick to pump any distance and the units would have to be placed at various locations in the mine to be workable.



		NOTE:	PICTORIAL MILL FLOW SHEET	THE ARTHUR W. WHITE MINE DICKENSON – SULLIVAN JOINT VENTURE

DICKENSON MINE

July 1983

THE MILL

HISTORY

The ore mined at Dickenson is one of the most complex gold bearing ores found in Canada. Gold is present in the ore in three different associations: Free Gold: Relatively large particles of gold that are not chemically attached to any other mineral. This type of gold is commonly referred to as V.G. or visible gold. Sulphide Associated Gold: Smaller particles of gold that are physically attached to the sulphide minerals contained in the ore such as pyrite, pyrrhotite and arsenopyrite. Sulphide Locked Gold: Extremely small particl of gold that are physically locked inside the crystal structure of the sulphide mineral arsenopyrite.

The original mill was constructed in 1948 and treated 125 T.P.D. The ore was crushed and ground to a fine sand, then the gold was extracted using cyanide. Recoveries were poor, averaging 75-80%. The mill circuit was unable to recover the sulphide locked gold, and was not recovering all of the free or visible gold.

In 1951, the mill was expanded to treat 350 T.P.D. A separate amalgamation circui was added to recover the free gold. A flotation circuit was added following the grinding circuit to recover both the sulphide locked and sulphide associated gold. The flotation concentrate was roasted in a new roasting plant with both the roaster calcine and flotation tailing being cyanided to extract the gold. Recoveries then averaged 90 to 93%.

In 1954 the mill throughput was increased to 430 T.P.D. and it quickly became apparent that the roaster was too small to handle all of the flotation concentrate being produced. The flotation circuit was then relocated to the tail end of the circuit to recove only the sulphide locked gold. Minor changes were made in 1959 with addition of a regrind circuit and a back fill plant.

In 1979 the roasting plant was shut down due to the environmental problems associated with the emissions from the plant.

In 1981 the mill was expanded to 700 T.P.D. by the addition of new grinding mill and a series of intensive aeration tanks designed to break down the sulphide locked gold so that it could be recovered by cyanidation. The expansion was never completed although the mill did run 700 T.P.D. for several months with recoveries averaging 78%. It was decided to go back to the drawing board and thus 1982 was used to plan the present expansion.

CRUSHING AND SCREENING

The Crushing and Screening plant at the mine is located in an extension on the nort side of the No. 1 Shaft headframe.

Mine run ore is hoisted to Surface and stored in a 31 foot diameter by 37 foot high circular steel bin having a live load capacity of approximately 900 tons. Ore is withdrawn from the bin and fed into an 18 x 36 inch Traylor Jaw Crusher, nominally set at 2 1/2 inches. The Jaw Crusher discharge is conveyed to a 6' x 14' Allis Chalmers double deck vibrating screen. The top deck is 3/4" slotted rubber screen with the oversize material being fed into a 4' diameter Symons standard Cone Crusher. The bottom deck is 1/2" slotted rubber screen with the oversized material being split between two three foot diameter Symons short-head cone crushers. Discharge from all three cone crushers is recirculated back through the vibrating screen.

The bottom deck undersize is the final crushed product and is conveyed through a 240 foot conveyor gallery into a 29 foot diameter by 29 foot high circular, woodstave

construction, fine ore bin. The fine ore bin has a live load capacity of approximately 700 tons.

Capacity of the crushing plant under ideal conditions is in the range of 65 tons per hour.

GRINDING AND CLASSIFICATION

Ore is drawn off the fine ore bin through three slot feeders and fed to two primary ball mills, a 6 foot diameter by 8 foot and a 7 foot diameter by 8 foot Allis Chalmers mills running in parallel circuit. Primary grinding is done in the presence of cyanide solution. Lime is added to maintain protective alkali ity and lead nitrate is added to counter the breakdown products of sulphide minerals present in the ore, which otherwise inhibit the cyanide dissolution of gold. The discharge from both primary mills is combined and pumped to a 7 foot diameter by 22 foot long secondary ball mill.

The discharge from the secondary mill passes across a 16" x 24" duplex mineral jig designed to recover the coarser particles of free gold from the ground ore slurry. The jig concentrate is collected in the jig box which is emptied once each day, and charged into a 36 inch diameter by 48 inch Denver amalgam barrel. Mercury is added to the charge and the mixture is ground for 6 hours.

Ground jig concentrates are emptied from the barrel and into a hydraulic cone. Amalgam and mercury particles, because of their high density, overcome the force of the rising water column, settle and are collected in a jar placed at the bottom of the cone. The lighter and finer pulps rise to the discharge point of the cone and are sluiced over a mercury coated, silver plated, copper table which collects any extraneous, non-settled, fine-sized mercury or amalgam. Discharge from the table is pumped back to the secondary mill for regrinding. Any mercury collected is re-cycled through the amalgamation circuit.

Amalgam collected in the jar is poured into a pail, allowed to settle, and any iron particles floating on the surface are removed with a magnet. The amalgam is pressed and sent to the refinery for further processing.

The overflow from the mineral jig is pumped to a 20 inch Krebs cyclone set in closed circuit with the secondary ball mill. Cyclone overflow is maintained at 40% solids to give a final grind size of 70% minus 200 mesh. The PH is maintained between 10.5 and 11.0. The cyclone overflow is pumped to two 30 foot diameter x 12 foot high double tray primary Thickeners arranged in parallel.

CYANIDATION AND PRECIPITATION

The cyanide circuit is essentially a two-stage straight cyanide process involving double agitation, thickening, filtering, washing and repulping. High-grade pregnant solution from the first stage is collected, clarified and sent to the precipitation circuit. Lower-grade solutions from the second stage are recirculated. Barren solutions are, for the most part, discarded.

The primary Thickener overflow, or pregnant solution is decanted by means of the overflow launders around the top periphery of the thickeners and flows to the clarifier tank.

The primary thickener underflow is pumped to two 28 foot diameter by 22 foot high Dorr style agitated wood stave tanks set in series.

Pulp enters along the top periphery of the first agitator, is drawn downwards, an raked towards the centre of the agitator and is then air-lifted up through the central shaf to the rotating sprayer arms which distribute the aerated pulp evenly over the surface. Th process is then repeated.

Calcium cyanide is added to the first agitator in sufficient amounts to bring pul titrations up to the range of 1 pound per ton. Fresh water is also added.

An overflow weir is located on the top periphery of the first agitator at a point immediately opposite to the intake. Excess pulp, containing relatively highly leached particles, overflows and enters the second agitator. Fresh water alone is added to this second agitator, and this only in amounts sufficient to lower the overflow density of the pulp to approximately 58% solids.

Air for agitation is provided at 30 p.s.i. to both units from a low pressure unit located near one end of the mill building.

Overflow from the second agitator flows by gravity into a splitter box and is pipe to two 11 foot 6 inch diameter by 12 foot Oliver drum filters arranged in parallel. The pulp is filtered, the cakes are washed with a water and are dropped into 4 foot by 3 foot by 16 foot paddle type repulpers located directly under each filter. The solution or primary filtrate is pumped to the clarifier tank.

Pure water is used for repulping, calcium cyanide is added to the repulpers in sufficient quantities to raise the cyanide strength to 0.4 pounds per ton, and the repulped cakes, at a pulp density of 58% solids, overflows into a pump box and is pumped to the first area of the second series of agitators. These, again, are all of the Dorr type. The first, No. 3, is 16 feet in diameter and 22 feet high. The remaining four are 14 feet in diameter and 16 feet in height.

Discharge from the final agitator flows by gravity into a splitter box, and is fed evenly into two, second stage, 28 foot diameter by 12 foot high, Dorr tray thickeners arranged in parallel. The secondary overflow (low-grade pregnant solution) is decanted off through the launders of the thickeners and flows to the grinding solution storage tank.

The second stage thickener underflows are pumped to two 11 foot 6 inch diameter by 12 foot long Dorr Oliver drum filters set in parallel. The slurry is filtered, washed an repulped with fresh water. The repulped filter cake is the final mill tailings product and transferred to the backfill plant to be either used as hydraulic fill underground or discharged to the tailings impoundment area.

High-grade pregnant solution, comprised of primary thickener overflow and primary filtrate, enters the concrete tank of the clarifier by first passing through a surge tank located at the front end of the clarifier. Levels in this surge tank are maintained by a pumping system arranged in closed-circuit with the previously mentioned grinding solution storage tank. In addition to any excess pregnant solution which is pumped from the clarifier the grinding solution is comprised of all of the secondary thickener overflow and some barren solution from the precipitate press. Because of it's overall low-grade nature, as little of it as possible is returned to the clarifier. Rather, it is used up in the secondary mill of the grinding circuit, and as a dilution in the jig.

The clarifier leaves consist of canvas-covered wood spline frames which are connected to piping leading to a Merrill-Crowe vacuum deoxidizer. The canvas is treated with

a filtering aid called Diatamaceouscarth. Solutions flow from the surge tank into the main clarifier tank and are then drawn through the canvas coverings up through the leaves and in the Merrill-Crowe vacuum tank, which is filled with cribbed wooden blocks to cascade the solution. Being under vacuum the solution is de-aerated, and drawn into a pump located at the bottom of the tank. Zinc dust at the approximate rate of 20 pounds per day is added fr a feeder, and the mixture is pumped to a 36 inch Crowe, frame leaf, precipitate press. Ti piping system between deoxidizer and press is 75 feet in length to ensure that a good distribution of zinc dust through the solution takes place.

Zinc enriched solution is forced through frame press leaves over which have been lapped one thickness of No. 10 duck canvas and two thicknesses of filter paper. Barren solution passes through the filtering medium and the solid precipitates remain between the leaves of the press. Three presses arranged in parallel are used. Presses are operated until the pressure on the intake line builds up to 35 p.s.i. They are dismantled and cleane usually every 15 days and the resulting precipitate is sent to the refinery.

Barren solutions from the press pass through a Lea recorder and are pumped to a barren solution storage tank. Due to its fouled nature - it contains relatively high amount of arsenic and antimony compounds - the bulk of this solution is discarded. The remainder is used as a gland wash for the S.R.L. pumps in the mill, and along with the secondary thickener overflow, as a grinding solution in the secondary mill. Some is also added to the mineral jig.

REFINING

Refining is carried out twice per month. The operation consists of melting, reducing and upgrading two products: amalgam from the amalgamation circuit and precipitate from the cyanidation circuit.

Amalgam is retorted in a small propane-fired, water cooled, apparatus and the mercury is driven off and recovered for further use. The sponge is collected for further treatment.

Precipitate is weighed into batches and mixed in a box with a flux consisting of the following:

Borax
Soda Ash
Fluorspar
Silica
Nitre

Two, oil and air-fired Rockwell type, furnaces are used for refining. These are brought up to a heat of 1800 to 2000°F and 150 pound batches of mixed precipitate are charged into each furnace. There are usually three of these batches at each refining. The burners are reignited and the charge is heated for approximately 2-3/4 hours or until no caking of the precipitate on the sides of the furnace is evident when the furnace is rolled.

The entire charge is then poured into button type, pot moulds, cooled and dumped. Slag is broken off the buttons and sent to Noranda for further processing. The metallic buttons, along with the amalgam retort sponge, is charged back into one of the furnaces. Enough flux to cover the bullion is added with the charge.

The charge is heated 45 to 60 minutes and poured under slag into 1000 ounce bar moulds. The moulds are cooled, dumped and the slag is separated from the bar and stored for reverting.

Bars are cleaned and brushed, sampled, weighed and stamped. They are then packed into aluminum boxes and shipped to the Royal Mint for further refining.

Approximately 4,000 ounces of gold are produced each month. Bars generally weigh the range of 1000 ounces and have a fineness of 820 gold and 80 silver. The bulk of the remaining material is copper.

MILL EXPANSION PROJECT

The present expansion is intended to increase the milling capacity from 500 to 700 T.P.D., a 40% increase. In more detail the following changes are being made to the mill:

CRUSHING PLANT- In the past the crushing plant has always been the obstacle to increasing the mill throughput. The crushing plant has historically been asked to produce a fine 1/4 inch sized gravel. To accomodate the increased tonnage the final product size is being increased from 1/4 to 3/4 inch sized gravel by replacing the sections of rubber screen deck. This will allow the crusher settings to be increased permitting the crushing plant to process more tons per hour. In effect, the crushing plant will be able to handle the higher tonnage by producing a coarser final product.

As part of the expansion project the dust collection system in the crushing plant is being overhauled. The system will provide downcasting ventilation through each of the crushers and will enclose most of the conveyor transfer points. Dust collected in the fabric dust collectors will be slurried and pumped over to the grinding circuit rather than returned to the screen as is present practice.

GRINDING CIRCUIT- Following the 1980 expansion, the grinding circuit had more installed horsepower (1,050 hp) than was required. Thus it was decided to more fully utilize this power by transferring the work load from the crushing plant to the grinding circuit. To handle the coarser crushed product, one of the grinding mills, an 8' diameter x 12' long Allis Chalmers mill, is being converted from a ball mill into a rod mill. This change required that the mill be taken out of service, stripped down and rebuilt. A new feed conveyor, raised operating floor and a mechanized rod charger are being installed. Instead of using 2 1/2 inch diameter steel grinding balls, the rod mill use approximately 40 tons of 1 1/2 diameter by 11 1/2" long steel rods to do the grinding. The other three grinding mills will be converted into secondary and tertiary grinding units designed to further reduce the size of the ore after passing through the rod mill.

A total of three jigs are to be installed in the new grinding circuit to recover free or visible gold contained in the ore.

CYANIDATION CIRCUIT- The present mill circuit has a retention time of 35 hours. This is the length of time the ground ore slurry is kept in the mill leaching tanks. To increase the mill throughput to 700 T.P.D. without adding new leach tanks would decrease the retention time to 21 hours and subsequently reduce the mill recovery of gold. To counter this problem a new 36 foot diameter by 24 foot high steel leach tank was erected. The agitator mechanism for this tank has a nine foot diameter turbine style propellor blade and is driven by a 50 hp drive motor. This one tank will hold over 150,000 gallons and increase

the circuit retention time to 48 hours at a mill throughput of 700 T.P.D.

MILL SERVICES- Over the years the piping in the mill has become unservicable. In addition our concern with the environment and how our mining operations affect this environment have increased. As part of the present mill expansion several changes are bein made to reduce the mill's consumption of fresh water. A new 28 foot diameter by 12 foot hi steel water tank has been erected outside the mill. The water used to cool the underground compressors will be pumped to this tank and will become the main supply of water for the mi In times of need, this compressor cooling water can be supplemented with water pumped out o the mine, and as a last resort, fresh water. In addition, new pumps and piping systems are being installed in the mill to permit the recycle of barren solution which is presently being sent to the tailings pond.

It is anticipated that these changes will have the mill treating 700 T.P.D. by August 01, 1983 with an average recovery of 85%. In the near future, we hope to improve this recovery to 90% by rebuilding the old flotation circuit and producing an arsenopyrite concentrate containing the sulphide locked gold, which will then be shipped by rail to a smelter.

NOTES OF MEETING WITH GEORGE HOPE, MACASSA MINE

MARCH 25/85

Macassa's mill superintendent is Len Robinson.

Mill feed presently comes from Macassa (350 TPD) and Lake Shore (150 TPD).

There are some differences in the geology of various ore bodies at Macassa but no real trend as the mine deepens. There is some molybdenum and lead telluride in the ore. With tellurides, a high lime level in cyanidation is desirable.

Maximum mill capacity is 500 tons/day.

The ore is ground to 95% - 325 mesh.

Macassa have a tower mill in the circuit. Maintenance, particularly on the flights, is quite high but the mill is needed in the circuit. It is believed to be 15 - 20% more efficient, power wise, than a comparable ball mill. Harder Ni-Hard alloys are being tested on the flights. Impact is not a problem on the flights.

The mill flowsheet is much as shown in Milling Practice in Canada 1978, but the tower mill has been added. Two preaeration agitators on the Unit 1 line should be shown between classification and thickening.

With the higher tonnage now, leaching is single stage at 45% solids, with a retention time of 50 - 60 hr. Filtration is apparently also single stage. (Soluble losses must be high.)

Approximate reagent consumptions are as follows:

	Lb./Ton
Lime	8
Calcium Cyanide	1.2 (as NACN equiv.)
Zinc	0.2
Lead Nitrate	0.002

Control levels for lime and cyanide in leaching are about 1 lb./ton of solution for each.

Most mill assays are run by atomic adsorption but Macassa relies on fire assays for the metallurgical balance.

Scavenging gold from the tailings by carbon adsorption has been tested but loadings are low because of dilution from mine water.

A new mill to treat ore from Lake Shore and/or Macassa is a possibility..

Hope is of the opinion that a meter to measure cyanide level in leach solutions would not be very useful because retention time is so long that cyanide control is no problem. However, he would like to see a meter developed to monitor gold levels in barren solution so prompt action could be taken to correct high barrens.

Lac Minerals staff were too busy to show Canmet's observer through any of their mills, viz. Macassa, Doyon, Terrains and Est. Malartic.